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Engineering and Mining Journal

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NEW YORK LONDON

THE Alaska Juneau Enterprise

An account of practice developed and equipment in use for mining and milling, with details of policy and organization, direction, and management at the world's premier low-grade gold mine

▼ ▼ ▼

FREDERICK W. BRADLEY, President, Alaska Juneau Gold Mining, Bunker Hill & Sullivan Mining & Concentrating, and other companies. President, American Institute of Mining & Metallurgical Engineers, 1929; W. L. Saunders medalist, 1932. Member, Mining & Metallurgical Society of America, Canadian Institute of Mining & Metallurgy, and the Institution of Mining & Metallurgy.

September, 1932

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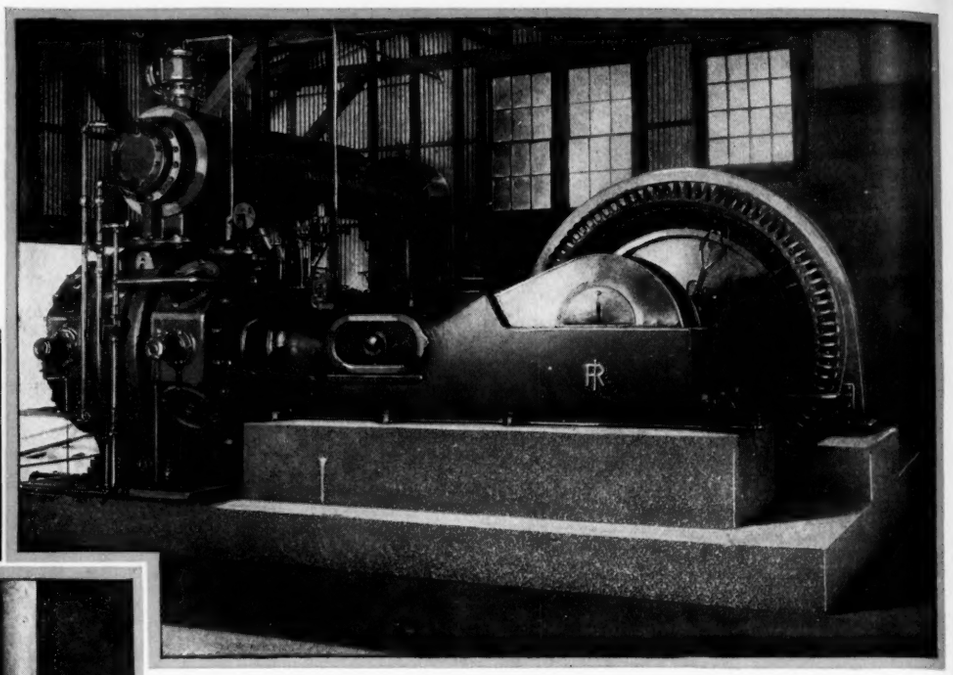

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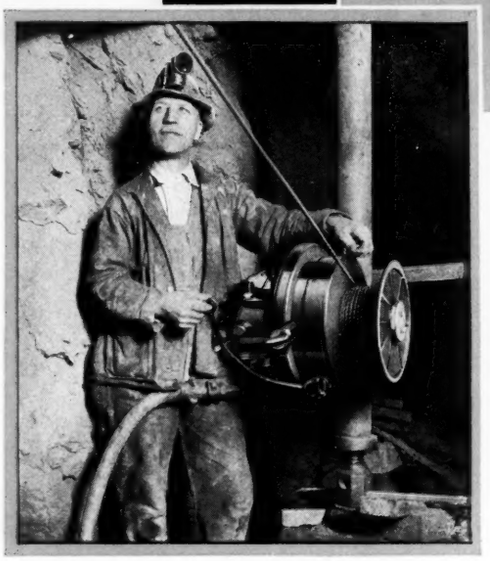

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Engineering and Mining Journal

with which is consolidated

Engineering & Mining World

A. W. ALLEN, *Editor*

September, 1932 Vol. 133—No. 9

THE ALASKA JUNEAU ENTERPRISE

DURING the past few decades of mining history the improbable has materialized into the actual, the unthinkable has become accepted fact. Improved facilities for the spreading of knowledge have played an important part in the valuation of effort in its responsibility for effect. As story after story reveals the economic achievements of the engineer, in places where valuable mineral is sparsely scattered through valueless gangue, the truth is being recognized that intellectual and personal integrity in the practice of knowledge and the utilization of skill is mainly responsible for the result. Such attributes, complemented by engineering acumen and persistence, and backed by sufficient capital, have effected and will again effect the transformation of a mineralized deposit of rock into a great national resource. . . . Mining engineers may be classed as timid or courageous. Those who invent, innovate, and pioneer in the highest branch of professional endeavor—who add substantially to the wealth of the world—must face a period, sometimes distressingly protracted, when their efforts are discouraged by skepticism. Happily, however, the croakings of the pessimists pass into oblivion in the new enthusiasm that haloes the achievements of those in whose ambitions they had professed no faith. . . . Pursuit of the inevitable, be it failure or success; the uncovering of fact, be it favorable or unfavorable—such venturings lead to acknowledgment of those sterling qualities of moral steadfastness and technical foresight that mark the characters of the outstanding engineering pioneers. No greater opportunity for the exercise of such qualities can be found than in the task of providing for the needs of present and future generations by adding to the resources of the world, by effecting the transformation of unpayable rock into profitable ore. For his contribution to work of this kind the name of Frederick Worthen Bradley is accorded a deservedly high place among the comparatively few great makers of the world's mineral reserves: the engineers of new mining enterprises. . . . The story of Alaska Juneau is told in the pages that follow. Difficulties that seemed insurmountable to those of little faith and courage would have caused failure but for the tenacity of purpose with which Mr. Bradley insisted that they be recognized, and overcome by technical endeavor of the highest quality. The articles, which describe each phase of well-coordinated methods for mining and ore dressing, also reflect credit to the heads of staff, particularly to Mr. Bradley's brother, Philip; also to Mr. L. H. Metzgar, the general superintendent of operations at Juneau, to the department heads and to subordinates in mine, plant, and office. . . . *Engineering and Mining Journal* is privileged to chronicle details of the history and technical direction of a gold-mining enterprise that owes its success to pioneering courage, wise leadership, and skillful supervision, rather than to record an example of the exploitation of a favored deposit of normal grade. Continued appreciation of the value of the ore developed at the Alaska Juneau mine would be a happy and deserved outcome in harmony with precedent established elsewhere, indicating that a large-scale production program under competent direction is a dominantly favorable factor in the making of a great property.

A. W. Allen

Alaska Juneau

Two prospectors, grubstaked by Sitka merchants, discovered the outcrop of the Juneau gold belt in 1880. Seventeen years later, following pioneer exploration and development of the deposit that led to the working of high-grade quartz veins below the decomposed outcrop, the Alaska Juneau Gold Mining Company was organized to exploit a group of lode claims that had been acquired by purchase. Opportunity for profitable operations soon became apparent. Two stamp mills were erected, and later a large, modern concentration plant. Costly errors in design and practice, complicated by management difficulties, subsequently required a wholesale program of alteration and reconstruction. Since those trying days, operations have improved year after year, the cost of milling gradually decreasing while the amount of gold recovered per ton of ore treated steadily increased. . . . Operations at Juneau are in charge of a general superintendent. Duties of each department head are clearly defined, with no overlapping of authority that might develop a "twilight" zone of responsibility. . . . Plans for future mining at depth are intimately associated with a post-mineral fault of major proportions, known as the Silver Bow fault. Shrinkage stoping will be used, with mining starting simultaneously from several levels. Ore in the pillars will eventually be extracted by caving. Haulage levels will be spaced 1,000 ft. apart. The design and specifications for the main hoist are being prepared. Skips, not cages, will be used.

P. R. Bradley
Consulting Engineer

History, Organization, and Outlook

Rumors having reached Sitka in 1880 that Indians had discovered gold on the mainland of Alaska, several Sitka merchants organized a small company and grubstaked two prospectors, Richard T. Harris and Joseph Juneau. These men made their first trip to the mainland in a small boat, and succeeded in finding gold in a creek (afterward named Gold Creek) that emptied into Gastineau Channel near the site subsequently selected for the town of Juneau. The two prospectors returned to Sitka, reprovisioned their boat, secured additional help, and returned to the place of their discovery. First attempts at mining in the bed of Gold Creek were unfruitful of result, so exploration was made of the higher reaches of the creek, and ultimately the outcrop of the Juneau gold belt was discovered in Silver Bow Basin, about two miles inland. Lode locations made immediately thereafter now constitute the principal area of the Alaska Juneau group of claims.

News of the gold discovery created a mild rush to the district in the following year, and the field soon became an active and producing camp. The decomposed outcrop and the creek immediately below the outcrop were both profitably mined by placer methods. This pioneer work led to the finding of high-grade quartz veins that could be worked profitably in arrastras. Mechanical grinding devices of a type in vogue then superseded arrastras, and in 1893 a twenty-stamp mill was erected. By that year the gold production of Alaska was estimated by the United States Geological Survey to have been $1,000,000 from lode mining, and $1,250,000 from placer mining.

In February, 1897, the Alaska Juneau Gold Mining Company was organized, and acquired 23 patented lode claims by purchase. Such claims now cover a continuous area of about 1,000 acres on the Juneau gold belt, extending for more than a mile along the outcrop. In the year of its organization the company erected on the outcrop the original five-stamp mill that had been used on Douglas Island for the original mill tests made on the outcrop of the Alaska Treadwell mine. This five-stamp mill and a 30-stamp mill built in 1896 were operated by the company on ore mined from open pits at selected spots on the outcrop, where quartz stringers were most numerous. The miners doing this pioneer work learned that the slate and gabbro between quartz stringers contained little or no gold, so this was rejected and sent to waste dumps; the quartz, as clean as it could be made after separation by hand, was the only material sent to the mill. So in the earliest days of milling on this deposit the necessity was recognized for sorting ore from waste rock. Facilities for sorting were included in the pilot mill built in 1913, but were excluded by the designers of the 8,000-ton mill erected in 1916; and this neglect was one of several errors in mill design that at a later date nearly wrecked the enterprise.

Between 1893 and 1914 the two stamp mills mentioned crushed 295,807 tons of ore selected out of 472,783 tons mined. These early operations were carried on in the summer only, as climatic conditions were too severe at that altitude to permit mining or milling in the winter. During the time this ore was being mined, sorted, and milled, a close study was made of local conditions and problems, and valuable economic and technical data were gathered. Information developed as a result of the study disclosed that the Juneau gold belt, an integral part of the Coast Range geology, is of great length, of variable width, and of irregular mineralization. The belt consists of slate and similar sedimentaries, with intrusions of metagabbro. It is divided into bands of poor definition, which converge in the vicinity of Juneau, becoming better defined and constituting a broad lode. Gold in the belt is due to the presence of quartz. In the vicinity of Juneau the broad lode contains an enormous tonnage of low-grade ore that could be mined at a commercial profit when operating on a large-scale all-year basis. Large-scale all-year operations demanded low-cost underground mining, sea-level mills, and a continuous supply of cheap power. Sea-level mills were necessary to secure all-year water for milling purposes, because the fresh water supply at higher levels would be frozen during the winter season.

As soon as chances for profitable operations became apparent under proper conditions the belt began to receive more serious attention. In 1898 the Alaska Juneau company started work on a deep-level adit from a point near the shore of Gastineau Channel. In 1910, F. W. Bradley entered into a contract with the Alaska Juneau company, under which he was to drive a tunnel from a point 420 ft. above sea level to a point beneath the surface workings. This tunnel was started in 1911 and completed in about two years. In 1912 the newly organized Alaska Gold Mines Company began a 10,000-ft. adit at a point 670 ft. above sea level. In 1912, United States Smelting, Refining & Mining Company acquired the Ebner group of claims, and started an adit from a point 350 ft. above sea level.

The tunnel, called the Gold Creek tunnel and driven under the terms of the F. W. Bradley contract with the Alaska Juneau company, followed the Silver Bow fault. This is a major fault with a general east-and-west strike, and a horizontal displacement of 2,100 ft. The vertical displacement has been estimated to be between 1,200 and 5,700 ft. In the mine records, that part of the mine lying on the north side of the fault is called the north orebody, and the segment to the south is called the south orebody. The effect of faulting on the belt was to move the segment on the north

side 2,000 ft. in a westerly direction, and to drop it 1,800 ft. vertically. The north orebody was encountered by the Gold Creek tunnel at a point about 4,000 ft. from its portal; the diagonal intersection was about 600 ft. long, with average assays of about $2 gold per ton. At a point about 6,500 ft. in, a two-compartment raise was driven to the surface, holing through at a point near the early surface workings. This work completed the first part of the F. W. Bradley contract. The second part was the construction of a mill, on the shore of Gastineau Channel, with a capacity of 23,000 tons per month.

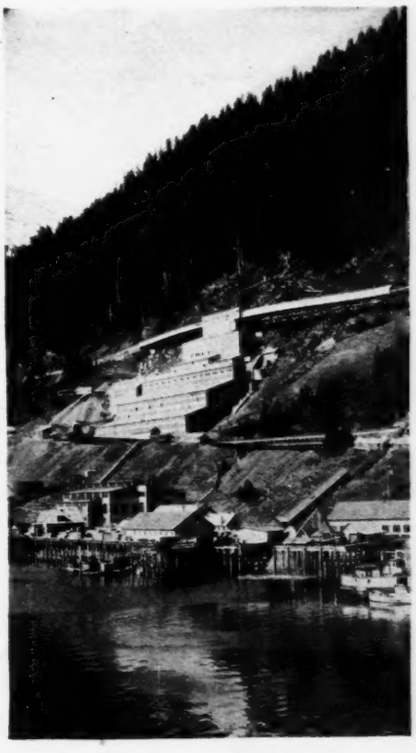
Many factors were considered in the design of this pilot mill, built in 1913. Sorting, proved necessary by the early operations, could not be practiced underground concurrently with any low-cost mining method; therefore, provision was made in the pilot mill for sorting out waste rock before and after coarse crushing. Stamps were selected for fine crushing because many years of experience at the neighboring Douglas Island mines had proved stamps the cheapest fine-crushing equipment then known. Stamping costs on Douglas Island were 10c. per ton. The intention was to increase the number of stamps in the pilot mill from the original 50 to the maximum number a standard crew could handle—150 stamps—as soon as the flowsheet was worked out satisfactorily, and as fast as finances permitted; then, with the earnings from this mill, the building of three additional similar units was planned. However, the 600 stamps in the four units would not, when operating under standard stamping practices, have a capacity to match the mine output, and the hillside provided space for four 150-stamp units only. Consequently, an increase in the stamp duty was planned by stamping to ¼ in. and finishing in secondary grinding equipment. The discharge from this equipment was to be classified, the overflow sent to waste and the underflow to amalgam plates, and thence to concentrating tables. The lower part of the mill was so designed that changes in the flowsheet could be made with ease and promptness, for the recovery of lead and iron sulphides separately was expected to require much experimental work with concentrating equipment of various types.

About this time the plans of the Alaska Gold Mines Company had taken definite form, and were beginning to wean Mr. Bradley's partners away from the abovementioned and agreed-upon program for Alaska Juneau financing and mill construction. A definite split soon developed, the partners holding for issuance of stock to the public, a single mill of large capacity, and for large-scale mining and milling. Mr. Bradley held to the original program, and in the disputes that followed his health became impaired, and he was compelled to abandon all work for six months. Direction of company affairs was given to J. H. Mackenzie, who was authorized by the board of directors "to do and perform all acts necessary to the development and equipment of the company's mine for the output of 8,000 tons of ore a day, and to further do and perform all acts necessary in connection with the design and erection of the 8,000-ton daily-capacity mill." Funds had been provided by public subscription, against Mr. Bradley's wishes.

The mill designed and erected under the foregoing authority was started April 1, 1917. At the outset, glaring defects and errors became apparent throughout, the most disastrous of these being the attempt to crush in one stage the discharge from the 9K gyratory crushers in an untried type of ball mill to pass 7-mesh screens. The mill failed sadly to perform according to expectations; the capacity proved less than 50 per cent of that promised, and the costs were over three times as high as anticipated. The company's funds were exhausted; the war, with attendant high wage and supply costs had arrived, and the outlook was dismal. Mr. Mackenzie resigned, and Mr. Bradley, who had now recovered his health, resumed control. A vigorous campaign of alteration and reconstruction was started, facilities for sorting out and disposal of waste rock were designed and fitted in as far as practicable in a steel and concrete structure; and capacity was gradually increased, costs were lowered and metallurgical efficiency was improved. The accompanying table illustrates the various steps of rehabilitation in terms of figures; but the human phase of the achievement—the financial struggle, the tenacity of purpose, the faith in outcome and loyalty to friends and business associates—is by far a larger story than any that can be told by figures. In the accompaning chart, the increase in tonnage, as well as the gross value recovered and various costs, are shown graphically.

The Alaska Juneau company is organized under the laws of West Virginia. In that State the charter tax is modest, and the laws pertaining to nonresident domestic corporations are free from burdensome features. The directors elected at the annual stockholders' meeting are to be found in or near the San Francisco office of the company, and are therefore available for consultation and meeting on short notice. The directors and the president have complete authority over all corporate affairs, matters of finance and policy, questions of active management in their broad phases, planning for development and for future activities, and the purchase and sale of property.

Next in order is the consulting engineer, whose duties, in a word, are those of a mobile contact agent between the board of directors, the president and the operating staff. More generally, these duties consist of keeping before the president in concise form, with recommendations, major managerial problems that may arise; changes that might be made to advantage in surface and underground equipment, changes in milling equipment or metallurgical treatment of



Statistics of Technical Progress

Year	No. Days in Operation	Tons Mined per Day	Per Ton Mined		
			Cost of Milling	Gold Assay Value	Gold Recovered
1917	274	2,472	$0.7171	$0.86	$0.64
1918	363	1,631	0.6762	0.92	0.73
1919	363	1,909	0.5608	0.99	0.72
1920	364	2,590	0.4463	1.08	0.78
1921	363	4,445	0.3554	0.86	0.60
1922	361	6,400	0.2613	0.81	0.56
1923	361	6,860	0.2662	0.81	0.58
1924	362	8,476	0.2512	0.87	0.62
1925	362	9,618	0.2504	0.85	0.58
1926	362	10,579	0.2154	0.75	0.50
1927	362	11,790	0.2257	0.77	0.55
1928	363	10,243	0.2285	1.11	0.84
1929	362	10,598	0.2319	1.12	0.89
1930	363	10,801	0.2271	1.10	0.86
1931	363	11,466	0.2160	1.12	0.89

ore, or in mining operations, any of which changes would involve capital expenditure; plans for the future that necessitate preparedness in all departments; legal questions that affect titles or other important matters; wages, welfare, legislation, and community contacts; and any important details that are beyond the routine duties and scope of the local staff.

Offices of the secretary-treasurer, the assistant treasurer, and the purchasing agent, as well as those of the chief counsel of the company, are also in San Francisco. These men perform the usual duties of their several offices under the general guidance of the president.

At Juneau, all company activities are in charge of the general superintendent. His control and supervision spreads over the entire operation through department heads, each in charge of a logical division of the work. Several fundamentals must be well considered and understood by all department heads if satisfactory results are to be secured: (1) Every man vested with authority must know who his boss is, and who he himself is to boss; (2) the difference between authority and cooperation must be clearly recognized and applied; (3) interference with work, arising out of clashing personalities, friction, and political maneuvering will not be tolerated. In addition, each department head must have his duties well defined and his jurisdiction established to avoid an overlapping of authority concerning which two department heads may clash, so that no "twilight," undefined duties may be neglected.

For the benefit of each department head, separately and for all collectively, an "Organization Flowsheet" was prepared; this defined the duties of each department head, and showed graphically his relation to all other departments, the lines of authority, and the lines of cooperation:

Mine Foreman—To have charge of underground and surface mining operations, including development, stoping, bulldozing, loading, tramming, hoisting; distribution and care of explosives in conformance with established regulations; care and maintenance of drill-sharpening shops and machine drill repair shop; to anticipate and be responsible for all necessary labor and supply wants.

Mill Superintendent—In charge of all crushing, milling, and sorting operations; to anticipate and be responsible for all necessary labor and supply wants; to keep accurate records showing details of all operations and the distribution of all values recovered or lost; to have charge of all metallurgical research work, and to make recommendations for changes in the flowsheet that will increase extraction and/or reduce costs.

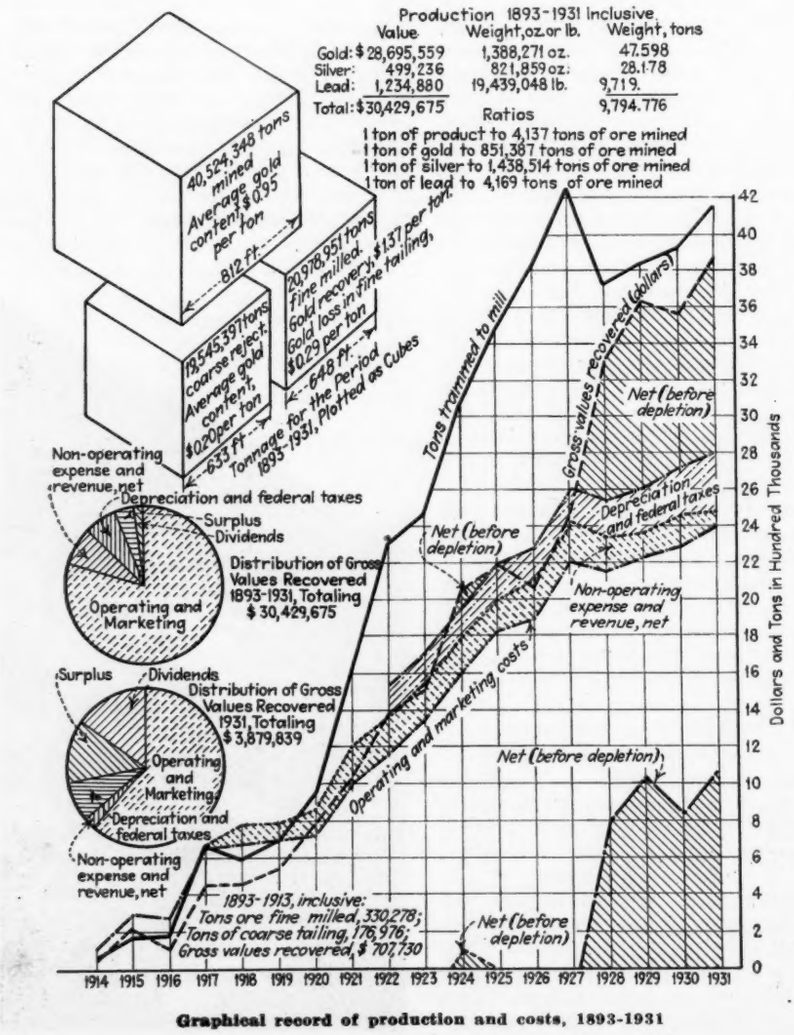
Master Mechanic—To be responsible for the upkeep and continuous smooth operation of all mechanical installation excepting electrical equipment; to have charge of machine shop, boiler shop, and general blacksmith shops; to be responsible for all steam heat, fire-fighting organizations and equipment, salt-water pump, change rooms; to deliver to the proper point all supplies issued upon requisition by any authorized person; to maintain all dams, ditches, flumes and pipe lines.

Boss Carpenter—To have charge of carpenter shop; to have charge of and be responsible for the repair and maintenance of all wooden structures; to anticipate and have on hand all supplies needed for routine and emergency repair work.

Chief Electrician—To be in charge of all electrical power-generating units, and all transmission, trolley and telephone lines; to maintain and keep in smooth operating condition trolley and storage-battery locomotives, motors, and other electrical equipment; to keep records of electrical energy generated and consumed, with details of consumption by departments and divisions; to anticipate supply needs in connection with the foregoing.

Chief Engineer—To have charge of surveys, patents, assessment work and property titles, plans, maps, data for technical records, recording underground geology; to prepare, file and be responsible for mine, mill, and other similar data and statistics for the monthly letter and other reports; to do designing, drafting, blueprinting and cost estimating of new construction work; to have charge of outside examination and prospecting.

Supply Department—To keep on hand and be responsible for as little stock as possible of all kinds, and to insure the continuous operation of all departments, mine, mill, and boarding house; to report defective supplies received, and to supply data for making claims therefor; to issue supplies on requisition signed by authorized persons only; to submit orders for supplies to the general super-



Graphical record of production and costs, 1893-1931

intendent for his approval; to receive and check against invoice all goods received.

Chief Clerk—To have charge of checks, payrolls, bookkeepers, timekeepers, stenographer, operating cost accounts, bank accounts, bullion and bullion records, insurance, taxes, collections, office files, documents, and document files.

Construction Work—One experienced foreman to be in charge of each construction job that has first been blueprinted, cost estimated, and approved by the general superintendent, the consulting engineer, and the president.

Boarding House—The boarding house and dormitory to be managed by a steward, without financial loss to the company and with satisfaction to the men.

Employment Office—To furnish men upon requisition of any department head or foreman; to keep labor data and records; to notify the doctor and the hospital of accidents, and to see that the injured secure proper care and attention from the doctor and hospital; to keep records of accidents and to settle, under the provisions of the compensation law, with injured employees.

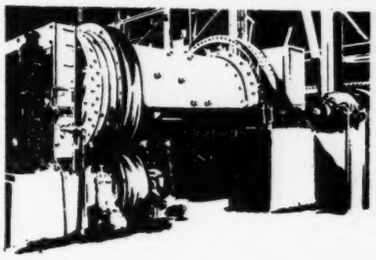
Doctor and Hospital—The doctor to give proper medical and surgical treatment to sick and injured employees, and report regularly thereon. The hospital to render proper care, attendance, and sustenance to employees in its charge, and to supply food, medicines, treatment, dressings and other necessaries prescribed by the doctor.

Committee of Safety—To investigate and report on accidents; to study and devise means of accident prevention, by establishing rules governing the work in all departments, which shall be put in force by the general superintendent; to maintain surface and underground first-aid crews, each to be brought under the direction of a duly authorized member of the committee.

Attorney—To have charge of legal matters in connection with the company's operations; to inspect, pass upon, and file for record deeds, agreements, contracts, and affidavits of assessment work; to perfect all patent papers so they will pass the land department without question; to take the necessary legal steps to protect the company's property against trespassers and to protect all rights and titles.

In addition to the foregoing individual instructions to department heads, the following general instructions were issued: All supply requirements must be properly anticipated, but with due consideration of the chance for accumulating obsolete stock; and all supplies must be purchased at no higher price per unit than is paid by any other buyer of equal volume and quality. No portion of the company's property can be sold or given away without the consent of the board of directors, and all the property must be protected against theft and trespass.

An organization flowsheet may be carefully prepared in great detail, but such a chart, with its definitions of duties, rights, and limitations, is of no consequence if the personnel is not loyal, efficient, and cooperative. The department heads of Alaska Juneau are working foremen, of high character, earnest, loyal, efficient, cooperating under the stimulus of pride in achievement. Their interest in the work and in the outcome of operations is all-absorbing. It binds them into a harmonious unit that is getting results.



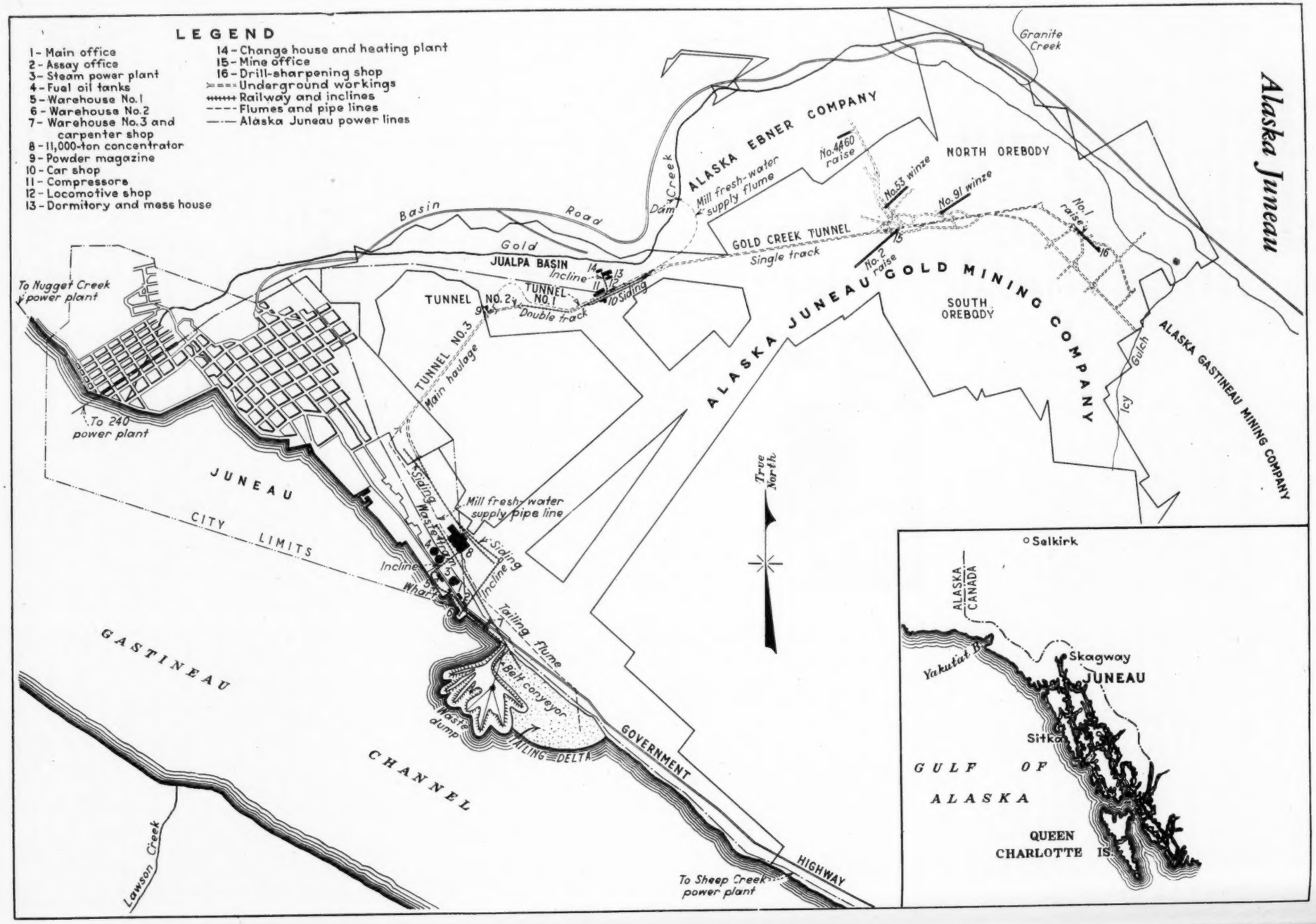
Future mining plans for Alaska Juneau are intimately associated with the most noticeable structural feature in the mine: a pre-mineral fault of major proportions that extends not only through the Alaska Juneau property but into the Alaska Gastineau property to the east and into the Dora, Ebner, and Hallum properties to the west. This fault, termed the Nugget Gulch fault in the mine records, is the footwall of the ore, though values have soaked into the underlying rocks at places. It has an average dip of just less than 60 deg. to the northeast, with variations from 50 to 70 deg., and was no doubt the main channel for the ascending solutions, which deposited the minerals that gave the adjoining country economic value. Sympathetic and subparallel faults may also have been vents through which the mineral-laden solutions ascended, but if the relative importance of an ore channel is measured by the increased degree of mineral precipitation along it, Nugget Gulch fault was the main source of mineral solutions. The 50 ft. of width adjacent to and on the hanging-wall side of the fault contains ore of higher grade than found elsewhere in the mine. Furthermore, this band of high-grade ore, as so far developed, is conspicuous throughout the Alaska Juneau and adjoining properties for its lateral and vertical extent.

Later than Nugget Gulch fault, and later than the ore, came the so-called Silver Bow fault, cutting through the Nugget Gulch fault and the ore, at an angle of about 45 deg., and dividing the mine into two parts. The horizontal and vertical components of throw of the Silver Bow fault are still debated by geologists who have studied the Alaska Juneau mine, and not until further work has been done will they be more in accord as to these measurements. Present estimates of the vertical throw on the plane of Silver Bow fault range from 1,200 to 5,700 ft. This point is of great interest at this particular time because of its bearing on estimates of the depth of the high-grade streak on the north side of the Silver Bow fault. Prospecting of this streak below the Gold Creek tunnel level was begun in May, 1930, by starting a small prospecting winze of one compartment at a point 50 ft. horizontally from the Nugget Gulch fault and driving it at an angle of 60 deg. from the horizontal. Owing to the fluctuations in the dip of Nugget Gulch fault, the prospect winze, between the collar and the 1,000-ft. level (vertical measurement), has been both under and over the fault at times, but the entire winze, to the 1,000-ft. level, measuring 1,201 ft. on the slope, has an average gold assay value of $4 per ton. The auxiliary prospecting work on the several levels, spaced about 150 ft. apart vertically, has shown the 50-ft. band of ore associated with Nugget Gulch fault to be higher in value in this section of the mine than elsewhere.

Additional work was planned and started, having in mind additional prospecting and the preparation of this band for stoping. The band itself was indicated by assays taken in 1913, when Gold Creek tunnel was driven through the ore on the north side of Silver Bow fault. At that time, 44 ft. of ore having a gold assay value of $6 per ton was intersected. Accordingly, as a part of the deep-level program, a second winze was started from a point 1,000 ft. easterly from the first and advantageously situated with reference to development work already done in the vicinity.

This winze has two compartments and a ladderway, and is being sunk to the 1,000-ft. level, where a connection will be made with the first winze.

Diamond drilling has been resorted to for exploring the country lying on the hanging-wall side of the high-grade band, and values sufficient for carrying on profitable operations are indicated, although the ore must all be hoisted. To be treated profitably, however, this ore on the hanging-wall side of the high-grade band will have to be broken and handled as cheaply as that above Gold Creek tunnel level. Accordingly, the high-grade band will be prepared for mining and stoped out first, this operation constituting the preparatory work for the lower-grade hanging-wall areas. To bring about further economies, main haulage levels will be 1,000 ft. apart vertically, and, in conformance with this arrangement, the


LEGEND
1 - Main office
2 - Assay office
3 - Steam power plant
4 - Fual oil tanks
5 - Warehouse No.1
6 - Warehouse No.2
7 - Warehouse No.3 and carpenter shop
8 - 11,000-ton concentrator
9 - Powder magazine
10 - Car shop
11 - Compressors
12 - Locomotive shop
13 - Dormitory and mess house
14 - Change house and heating plant
15 - Mine office
16 - Drill-sharpening shop
Underground workings
Railway and inclines
Flumes and pipe lines
Alaska Juneau power lines
Alaska Juneau
Granite Creek
ALASKA EBNER COMPANY
No.4460 raise
NORTH OREBODY
No.53 winze
No.91 winze
No.1 raise
Basin Road
Dam
Creek
Mill fresh-water supply flume
GOLD CREEK TUNNEL
Single track
No.2 raise
Gold
JUALPA BASIN
Incline
TUNNEL NO.2
TUNNEL NO.1
Double track
10 Siding
To Nugget Creek power plant
TUNNEL NO.3
Main haulage
ALASKA JUNEAU GOLD MINING COMPANY
SOUTH OREBODY
Icy
Gulch
ALASKA GASTINEAU MINING COMPANY
To 240 power plant
JUNEAU
CITY LIMITS
Siding
Mill fresh-water supply pipe line
Incline
Wharf
Tailing flume
Belt conveyor
Waste dump
TAILING DELTA
GOVERNMENT
GASTINEAU
CHANNEL
Lawson Creek
HIGHWAY
To Sheep Creek power plant
True North
Selkirk
ALASKA
CANADA
Yakutat B.
Skagway
JUNEAU
Sitka
GULF OF ALASKA
QUEEN CHARLOTTE IS.

1,000-ft. level will be made the first main haulage level. The usual intermediate levels will be driven for access, service, and convenience, but ore will be trammed to the main shaft on the even thousand-foot levels only.

For stoping, the shrinkage method will be used, and stopes will be started simultaneously from several levels in the block between the 1,000-ft. level and Gold Creek tunnel. Broken ore will be passed to the main haulage level either from stope to stope or through transfers. Dimensions of the stopes and of the pillars have not been finally decided upon, but, from what is now known, stopes 50 ft. long by 50 ft. wide, with 50-ft. pillars intervening, would seem to meet requirements best. The bottom of the first stope will be at a point about 50 ft. above the 1,000-ft. level, and the top of the upper stope in this block will be 100 ft. below the Gold Creek tunnel level, leaving that level protected by a sheet pillar 100 ft. thick. Ore to be broken in the stopes will therefore be much less than 50 per cent of the total volume, but, eventually, the pillars will be blasted and the entire mass allowed to cave along with the lower-grade hanging-wall areas. Thereafter, operations will be quite similar to the present methods used in mining the ore above Gold Creek tunnel level. By following this program, ore remaining in the pillars will eventually be mined and delivered to the shaft at a cost less than that for mining ore in the shrinkage stopes.

Much study was given to the hauling of ore to the main shaft, handling of ore at the shaft, and the transfer of ore at the top of the shaft for transport to the mill. To obtain the cheapest stoping costs, rock should be handled in large pieces, but if these pieces are too large, additional expense is incurred in tramming and transferring at the shaft. Caging versus skip hoisting was considered, but a shaft of sufficient dimensions to handle 10-ton ore cars, standard mine equipment, was deemed altogether too large and expensive. Underground crushing in connection with skip hoisting was also given consideration, and rejected because of the expense involved.

The aforesaid subjects were viewed in terms of Alaska Juneau experience in handling large pieces of ore through ore passes, through chutes, into cars, and to the mill. This experience has perhaps been unique, and has resulted in the development of equipment and a system of handling that are somewhat out of the ordinary. Rock is delivered to the ore pass through grizzlies, with bars spaced at 27 in. on centers. Chute-gate equipment developed at the mine worked out so satisfactorily that a decision was made to use standard skip-hoisting methods, and to rely on locally developed methods for handling large material, thus saving the expense of an underground crushing installation.

Main-hoist design and specifications are being studied, but no details have as yet been developed to the point where they would provide material of interest for publication.

All plans and cost estimates for mining deep-level ores in the Alaska Juneau mine have been made after a close study of mining practices, costs, and experience at the Alaska Treadwell mine, on Douglas Island. The two properties are similar in general aspects, but present-day Alaska Juneau methods have many advantages over the Alaska Treadwell operations, which ceased fifteen years ago. A description of the differences that affect costs, capacities, and output follows:

At Alaska Treadwell the haulage levels were spaced 150 ft. apart and each level was a separate mining operation. At Alaska Juneau, the haulage levels will be spaced 1,000 ft. apart, with intermediate levels for service only. In Alaska Treadwell shrinkage stopes, drilling was with two-man piston drills; the ore was bulldozed or sledged in the stopes down to a permissible size for passing through loading chutes into 1¼-ton cars. These cars were dumped at the shaft station on flat grizzlies spaced at 12 in., where the ore was again sledged to a size that would facilitate handling into and out of skips. At Alaska Juneau the drilling will be done by modern one-man drills that will have a greater capacity than drills at Treadwell, as Alaska Juneau rock is more easily drilled. No sledging and no bulldozing will be done in the stopes. Ore will be drawn from the stopes through bulldozing chambers, where the grizzly bars will be spaced at 27 in. on centers. It will be reduced in size by passing through long ore transfers. Ore cars will be of 10-ton capacity. No secondary screening will be employed at the shaft ahead of skip loading.

At Alaska Treadwell the ore was hoisted by steam-driven, non-condensing hoists, whereas at Alaska Juneau the hoist will be of a modern electric type, driven by cheap hydro-electric power.

A comparison of the average Alaska Treadwell costs with the estimated deep-level costs for Alaska Juneau is presented in the accompanying table, in which the figures for actual and estimated costs are based on a daily average output of 2,280 tons per day at Alaska Treadwell, and an estimated output of 6,000 tons per day at Alaska Juneau.

Comparison of Average Costs

	Alaska Treadwell Costs, Cents per Ton	Alaska Juneau Costs, Cents per Ton
Development:		
Labor	8.6	3.0
Explosives	1.8	0.5
Supplies and power	8.2	1.5
Totals	18.6	5.0
Hoisting:		
Labor	2.0	2
Power	3.1	2
Supplies	4.4	2
Totals	9.5	6
Pumping:		
Labor	0.5	0.3
Power	0.4	0.4
Supplies	0.6	0.3
Totals	1.5	1.0
Stoping and tramming:		
Labor	31.6	25
Explosives	14.9	10
Supplies	9.5	8
Totals	56.0	43
Grand total	85.6	55.0

Alaska Juneau -

A caving system of mining is employed, with an ore transfer and four branch raises serving each of the large rectangular stopes that extend across the orebodies. Although experience gained in other mines of the district guided the development of the ore-extraction method, its character was primarily influenced by the necessity for low costs to insure profitable operations. Generally satisfactory ground conditions and wide orebodies aid materially in meeting this requirement. No timbering is used, except where faults are encountered, and the hardness of the ground enables it to stand up well against the wear from broken ore passing through drawholes and ore transfers. A sloping footwall, from which the ore breaks freely, provides a favorable starting point for stoping operations. Minor fault slips and quartz stringers also form planes of weakness that assist in fracturing large pieces of caved ground. . . . For the tonnage mined, the amount of development work performed is small. Stoped rock is trammed only on the adit level, none of the upper levels being equipped for haulage. Ore transfers are driven as development progresses, and when a drift or crosscut is near a connection with a chute raise, driving of the heading is stopped until the connection is made. Hand tramming of muck broken in development is consequently kept at a minimum. . . . Operations have been so arranged, and equipment so selected, that they will not be adversely affected by normal seasonal variations.

L. H. Metzgar
General Superintendent

Development, Mining, and Transportation

The Alaska Juneau mine is developed through the Gold Creek adit started on and driven along the Silver Bow fault—a post-mineral fault that displaces the north orebody, and separates the mine into the north orebody and the south orebody.

For the first 4,000 ft., or until the north orebody is reached, this tunnel is 10 ft. wide and 9 ft. high, accommodating a 30-in. gage single track and ditch. From the point where the orebody was first struck, development work has proceeded until parallel tracks through two adjacent tunnels, cross-connected in several places, now serve the mine from the 4,000-ft. point to the boundary of the property.

Inasmuch as the adit tunnel was driven along the Silver Bow fault, which crosses the ore at a horizontal angle of 53 deg., the tunnel crosscuts the north orebody diagonally. Good values were disclosed where the tunnel crossed the ore, and drifts were run on the adit level to develop the north orebody. At the same time a ventilating and working raise was driven in the footwall of the north orebody, and several levels were driven to prospect the orebody and provide facilities for working.

The main adit was continued through the north orebody and into the hanging wall of the south orebody, where No. 1 main raise was driven and connected with the surface to provide ventilation. Crosscutting on the adit level indicated that the south orebody was not payable on this level. Much ore was in evidence on the surface, so two levels were driven from No. 1 raise to find the bottom of the ore in this locality, and provide a base for development and mining. When these levels had roughly outlined the lower limits of payable ore, transfers were driven from the adit tunnel level to the No. 2 and No. 3 levels, to provide outlets. Depth of the adit from the surface where the orebody outcrops varies with the contour of the mountain, and ranges from 1,200 to 2,000 ft.

Development of the north orebody from the adit level is through 7x7-ft. raises that extend to an intermediate level driven 50 ft. above the main level while the raises were being driven. Crosscuts on this intermediate level are run to connect with the chute raises, bulldozing chambers are blasted out, and locations for the grizzlies are determined.

In the south orebody, where the pay ore is not indicated on the adit level, development has proceeded from the drifts and crosscuts driven from the main raise. When the extent and value of a block of ground have been roughly outlined by work on these levels, ore transfers are driven to the level indicative of the approximate bottom limit of pay ore in the vicinity. Branch raises are driven from the transfer, and the necessary connecting drifts and crosscuts are extended. The initial long transfers had ten branch raises, each with its separate grizzly. More recent ones have been shorter, with four branches only; better spacing of the grizzlies is possible with fewer branch raises. To provide many grizzlies tributary to one chute does not indicate desirable practice.

Development work required in proportion to the tons mined is small in amount, and no stoped rock is trammed on any level except the adit level, so none of the upper levels have been equipped for haulage. Intermediate drifts and crosscuts are only of sufficient size to be readily driven and to permit transportation of tools and explosives. The ore transfers closely follow development work, so the distance for hand tramming of muck broken in development is short. As soon as the drift or crosscut reaches the locality of its connection with a chute raise, the heading is stopped until a connection with a chute is made. The result is that hand tramming is seldom necessary for more than a moderate distance.

The north orebody below Gold Creek tunnel level is now being prospected and developed by two incline winzes, both of which will be sunk to a level 1,000 ft. below Gold Creek tunnel, where a haulage level will be established and preparations begun for stoping. Simultaneously, development work is proceeding on two intermediate levels, to outline the orebody. When this has progressed sufficiently, a large main working shaft will be located, sunk, and equipped to handle the tonnage developed. Meanwhile, development work and stoping will be accomplished through the two incline winzes, one of which has a single hoisting compartment equipped with a 2-ton skip; the other will have two hoisting compartments that will be equipped with 3-ton skips.

Although the growth of the method of mining practiced was guided by experience in other mines in the vicinity, the method adopted took into consideration ground conditions in the mine and the fact that low costs were necessary to insure a profit. The ground stands well, and no timbering is required except in places where the openings encounter faults.

Because of the fact that the ground stands well in stope extraction and preparation, and resists the wear of the broken rock through drawholes and ore transfers, the method adopted is practicable.

The plan of mining followed influences development work. For the purpose of outlining the mining method, a typical ore pass with four chute raises, and the preparation of the block of ground tributary to this ore pass, will be described: The four-branch raises are driven from the transfer so that when they connect with the bulldozing

level they will be 75 ft. apart along the strike and 100 ft. apart across the strike. The result is that the four grizzlies are so positioned that they form a rectangle, and serve a rectangular stope cutout 130 ft. along the strike and 180 ft. across the strike. When a branch raise and the bulldozing level are connected, a station is cut and the grizzly set. Some variation is made in the placing of a grizzly where necessary, to avoid setting it in weak ground. Great care is taken in making the connection and in cutting out the station, so that the grizzly will have a good bearing on solid rock.

The grizzly, in use for several years, is made up of three 16-ft. standard 15-in., 99-lb., steel girder beams. However, for the last two years a standard 15-in., 127-lb., H-beam, 16 ft. long, has been used with satisfactory results. Its design is shown in Fig. 1. The wide flange is stiffened on both sides of the web by cast-steel fillers, spaced at 39-in. centers on each of the center bars and on the inside of the side bars. A 1x15-in. plate is riveted to the top, which, besides being a wearing plate, makes the spacing between beams less at the top than at the bottom, which is desirable. The opening between the girders is 25 in.

Beams are set on a slope of 1¼ in. per foot (about 6 deg.) on solid rock, the lower ends being braced against the wall. Wooden blocking, to maintain the spacing, is wedged between the beams at both ends. Usually the center girder, which receives most of the wear, will handle over 100,000 tons of rock before it needs to be replaced. A worn center beam, when replaced, will generally render service as a side beam.

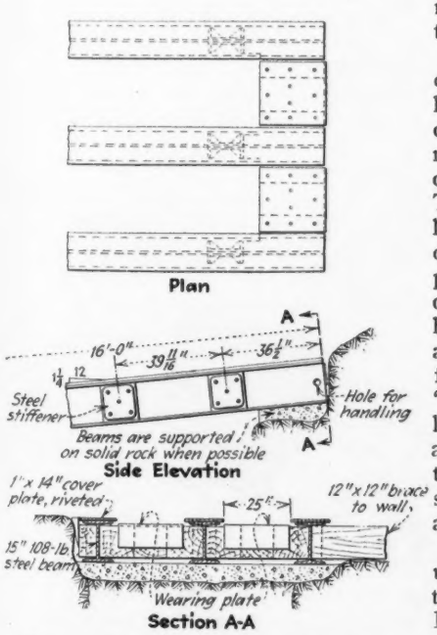


Fig. 1—Grizzly used in bulldozing chambers

When the back of the drawhole from the stope wears so that rock drops vertically onto the grizzly, the beams are reset in a position that is relatively higher and farther away from the opening. The bottom of the draw then fills in with fine material up to the angle of repose, whereupon the drawhole becomes as good as new. The nature of the ground determines the frequency of resetting the grizzly. Experience shows that, on the average, over 250,000 tons is handled before resetting is needed. The plan and section of a bulldozing chamber are illustrated in Fig. 2.

After the grizzly is placed, a 7x8-ft. raise is started from the upper end of the grizzly at an angle of 38 deg. It is carefully proportioned as to size and relation to the grizzly, afterward becoming what is locally called the drawhole, being extended for 18 ft. Shaping of this drawhole and its relation to the grizzlies must be carefully planned. A properly shaped and located drawhole facilitates production, whereas an improperly constructed grizzly and poorly shaped drawhole retard production. From this point four branch raises, called cutout raises, are started to outline the funnel-shaped stope cutout. These are driven on a slope of 38 deg., diverging at an angle of 90 deg. from each other. Those driven toward the outer limits of the cutout are extended until the limits of the cutout are reached. Those driven toward the adjoining grizzly are extended until they connect with each other. The result is that the cutout is outlined across the strike and along the strike by a set of raises having the form of a large letter "W."

When these raises are completed, cutting out begins just above the drawhole, and raises are started from the cutout to the level above, where connections are made with drifts or crosscuts previously driven to outline the ore. These raises provide ventilation and later serve as a base from which to drive powder drifts. Cutting out proceeds from the drawhole at a slope of 38 deg., the cutout from each drawhole assuming the shape of a funnel and proceeding until the four adjacent funnels meet at what is called the "peak" of the cutout, and its outer limits are reached. Successive stages are shown in Figs. 3 and 3A. Longitudinal section and plan of stopes in the south orebody are shown in Figs. 4 and 5.

The rectangular stope cutout measures horizontally about 130 ft. along the strike and 180 ft. across the strike. In the stopes on the footwall, the distance across the strike increases as the stope progresses upward. As cutting

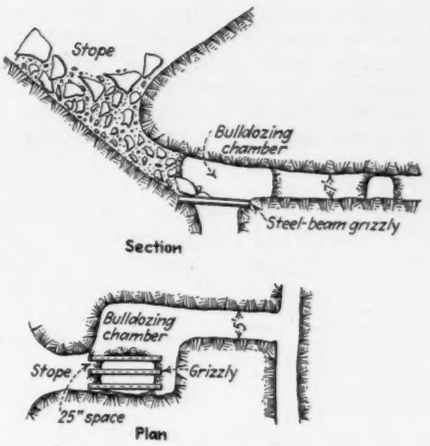


Fig. 2—Bulldozing chamber showing steel grizzly in place

out proceeds, some small supporting pillars are left, to steady the back. Cutting out is done in a retreating manner, access when the work begins being through an adjacent drawhole; but, as the work progresses, access to the stope is through a manway raise driven in the solid ground adjoining the cutout. When the cutting out is completed, the small supporting pillars are blasted, and breaking down the back with powder drifts begins. Cutting out is by contract at from 35 to 38c. per square foot, measured on the cutout slope of 38 deg. Contractors are required to keep the cutout 7 ft. high, and this results in a unit cost for labor and explosives of from 8 to 9c. per cubic foot for stope undercutting. Ingersoll-Rand No. N-72 drifters equipped with anvil block chuck and 1⅛-in. hexagon hollow steel are used in cutting out. Eight holes, of an average depth of 7 ft., are drilled per machine-drill shift; 45 sq.ft. per machine-drill shift is an average break.

The relation of development and preparatory mining costs to other mining costs is shown in Table I; the cost of development and preparatory mining in typical south orebody stopes, in Table II.

While cutting out is in progress, the broken ore is drawn from the stope to a sufficient extent to give room for work. Any heavy pieces of ground are blasted down; but, from the time cutting out begins, no necessity arises for crossing through the center of the stope. Access to the work is available from the side of the stope and along the pillars, and the work is conducted while retreating from the drawhole. Before cutting out is completed, the stope raises are connected to the level above. As cutting out nears completion, some powder drifts are driven. These are 4x3 ft. in cross-section, which is as small a space as is possible to work in. A maximum amount of the last several

rounds of muck is left in the drift for tamping when the powder is placed. All these powder drifts are carefully planned by the engineering department, and their progress is checked frequently, to insure selection of the proper location for placing the explosive. Each succeeding series of shots is laid out, and the powder drifts are driven, after the previous shots have been fired. Experience has proved this to be the most economical method.

Table I—Underground Cost per Ton of Rock From Typical Stopes in South Orebody

	Block No. 3	Block No. 4	Block No. 6
Development and preparatory mining	$0.0338	$0.0287	$0.0153
Stoping, powder in powder drifts	0.0062	0.0078	0.0016
Bulldozing	0.1230	0.1120	0.1175
Tramming	0.1000	0.1000	0.1000
Total underground operating cost per ton	$0.2630	$0.2485	$0.2344

Sometimes a shot will consist of a single pile of explosive, but more often it is of several piles. These are detonated in rotation or simultaneously, as required. If by rotation, great care is required to prevent destruction by the first shot of the connections of the succeeding shots. The amount of explosive needed for breaking is taken into consideration when the powder drift is planned. When the time comes for loading, the amount of explosive and its position are again discussed by the engineering staff and mine supervisors, after considering further information obtained while the powder drift was being driven. The amount to be loaded in each pile is then decided. No. 1 wire-countered Cordeau, with No. 8 caps and fuse, are used to detonate. No one enters the stope opening after first powder drifts are blasted, access to them being down through the stope raise from the level above. The grizzlies below are then actively worked, and the broken rock is drawn down, so that the rock broken by successive powder drifts will have some distance to fall, thereby causing further fracture of the ore.

As the stope progresses upward, its area is increased, because the powder drifts break to the footwall slip, which stands at an angle of 60 deg. When the back of the stope has progressed upward sufficiently, powder drifts are also driven alongside the opening, to enlarge further the stope and to induce caving. The stope is worked in this

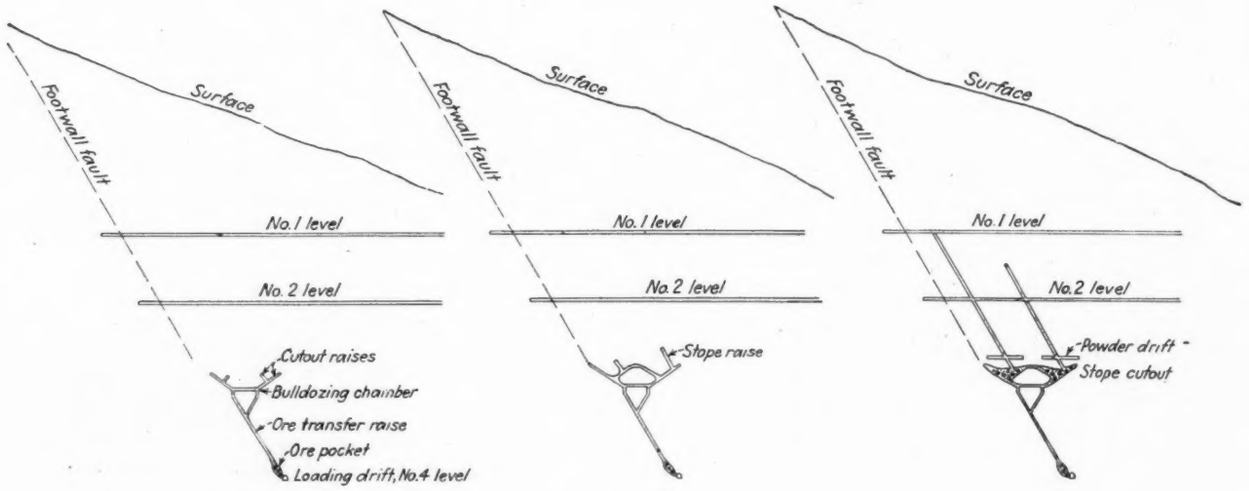


Fig. 3—First three stages in preparation and cutting of stope in south orebody

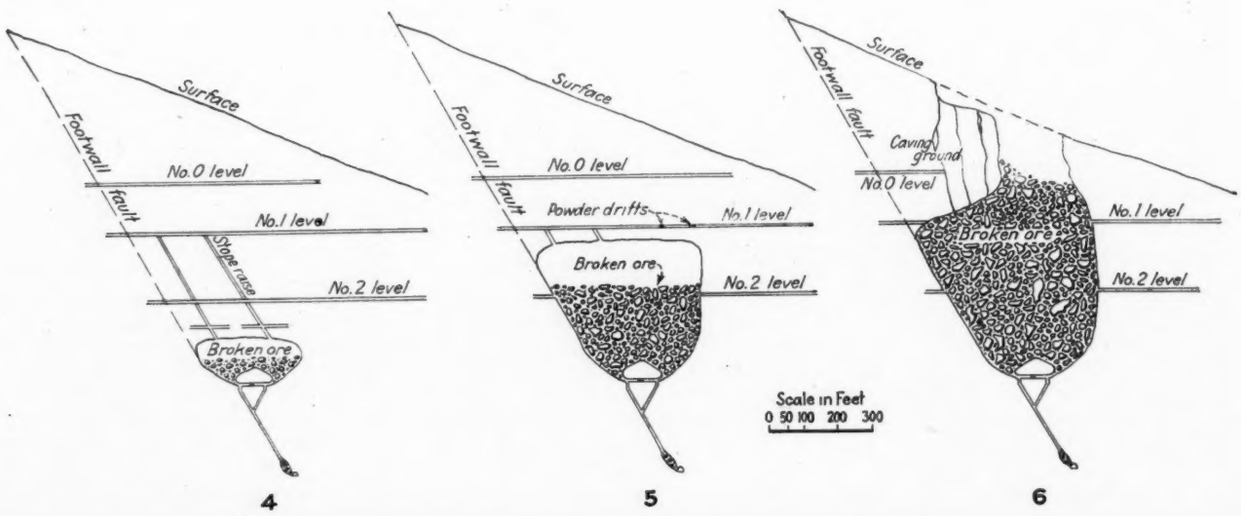


Fig. 3A—Last three stages in preparation and cutting out of stope in south orebody

Table II—Cost of Development and Preparatory Mining in Typical Stopes in South Orebody

	Block No. 3	Block No. 4	Block No. 6
Ultimate tons...	7,500,000	6,000,000	4,000,000
Tons drawn to date.....	5,906,880	4,532,110	1,529,530
Stope Development			
Ore transfer, ft.	2,850	1,307	616
Drifts and crosscuts, ft.....	4,207	3,424	480
Cutout raises, ft.	2,004	1,039	747
Manway and stope raises, ft.	2,514	2,611	604
Stope cutout, sq.ft.....	84,950	49,823	36,277
Powder drifts, ft.	3,230	2,878	559
Unit cost per foot			
Ore transfer....	$18.65	$16.68	$16.58
Drifts and crosscuts.....	14.18	13.91	13.08
Cutout raises....	12.68	12.40	12.44
Manway and stope raises...	15.65	12.39	12.44
Stope cutout....	0.56	0.63	0.63
Powder drifts...	8.84	9.09	9.12
Total development and preparatory mining cost	$253,735.59	$172,223.56	$61,252.96
Total cost per ton, ultimate	$0.0338	$0.0287	$0.0153

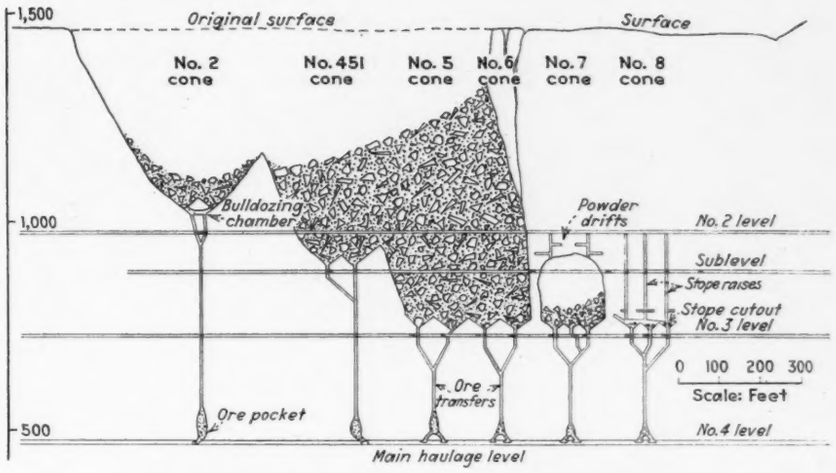


Fig. 4—Longitudinal section of stopes in south orebody

manner until the back has reached such a height and such an area that caving without blasting begins.

In placing powder drifts in the back and sides of the stope, the size of the opening and any slips or faults that may form the boundary of or cross the stope are taken into consideration. When the stope is first begun no difficulty is experienced in placing the powder drifts where they have a definite amount of ground on them. Later, although each has been carefully surveyed and a record kept of its location, no one can tell at all times how well the shots have broken or whether or not an overbreak has occurred. Fig. 6 is a plan and cross-section of a typical powder drift shot in the back of a stope. Fig. 7 shows the arrangement used in the west end of No. 3 cone.

The stope is not accessible, and determination of the shape of the back of the stope opening is not always possible, so a light Mitchell diamond drill has at times been used to drill through to the stope, to determine the distance from powder drifts to stope opening. With this information, the piles of powder can be accurately placed in relation to the stope opening.

The largest amount of rock broken by a single blast results when the pillar between the stope and an adjacent stope opening has been blasted, thereby causing extensive caving, which in some instances has extended through to the surface. In such event some of the caved material is in very large blocks. As the broken rock is drawn from below, these large blocks settle, and their weight causes them to break up and come down, so that they can be reached from the drawhole. Sometimes several holes must be drilled with a stoper in the large blocks that come to the drawhole.

As before mentioned, the grizzlies, placed before cutting out begins, are used from the beginning to draw the broken muck from the stope into the ore transfer. They are so placed that the free openings are 25 in. at the end under the drawhole, and slightly more at the end farthest from the drawhole. Any rock that can be barred through the opening easier than it can be blockholed is barred into the ore transfer. Generally, the ore passes are partly empty; and the fall of the large pieces breaks them to such an extent that little difficulty is experienced in drawing them through the chutes.

The powder ratio for secondary breaking varies with the age of the stope and the nature of the ground. During the first year of operation of a new stope in hard and tough rock, 3 tons per pound of powder is not an unusual ratio; whereas in the older slate stopes, 5.20 tons of rock per pound is ordinary. Average figures are 4.20 tons per pound of explosive consumed for secondary breaking.

The method of mining followed is made possible by the fact that the orebodies to be mined have great width. The ground stands well, and no supporting timber is required. Care is taken to position grizzlies and draw-

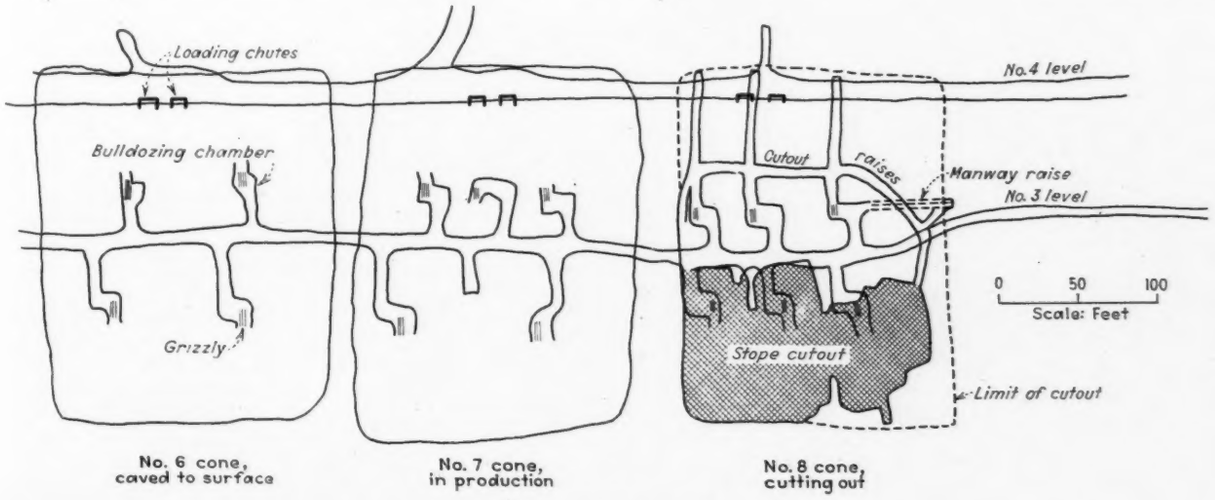


Fig. 5—Plan of stopes in south orebody

holes away from any fault slips, where the ground would rapidly be worn away by passage of broken ore. Conditions favoring the mining method are a footwall slip standing at an angle of 60 deg., from which the ore breaks freely. The first stopes were started along this slip, and from them the system of stopes has been extended. Some lesser fault slips also pass through the orebodies, and the quartz stringers that carry the values form in many cases planes of weakness along which breaking occurs. These quartz stringers are an important factor in the fracture of the large pieces of caved ore.

Gold Creek tunnel is driven along Silver Bow fault. Where the gouge is narrow and the rock much crushed by faulting, the tunnel is not timbered. It is 10 ft. wide and 9 ft. high. Where the ground has been timbered, it varies from 10 to 12 ft. wide and is 8 ft. high. On levels other than the main adit level a negligible amount of timbering is necessary. In cross-section the levels vary from 5x7 ft. to 7x7 ft., depending on the ground and the amount of work to be done on the level.

The ore transfers are untimbered, and driven 7x7 ft., the accompanying manway being 5x5 ft. These raises are connected at from 50- to 70-ft. intervals by a branch from the ore transfer 4 ft. wide by 5 ft. high. Stope raises are driven 5x5 ft.; powder drifts, 3x4 ft., or as nearly so as conditions permit.

No. 53 winze has a hoisting compartment 4½x4½ ft. in the clear, with a manway compartment 4½x3 ft. in the clear. Both are being sunk below the adit level. Ordinarily, drilling and blasting are done on one shift. Mucking, by hand, is done on the following shift. In the winzes and other faces worked on the three-shift basis, special ventilation is provided by means of 10-in. exhausting fans. No timbering is done except in the winzes. In each of these winzes the three bottom men per shift do both drilling and mucking. Details of winze sinking operations are given in Table IV.

No underground stations have required support. The principal underground structures aside from the grizzlies already described are the chutes and gates for loading cars. Where the chute raises are short, the chute is built of fir, using 12x14 in. timber for posts and correspondingly heavy timber for the remainder of the chute. Passage of ore is controlled by an underslung arc gate, 6 ft. wide. A check gate, consisting of pieces of 6-in. shafting, is hung below the head block, to prevent pieces of ore from flying out of the chute above the gate.

On the longer ore transfers, greatly strengthened and improved structures, built of steel and concrete, have been developed. Inasmuch as, in each instance, a large tonnage is to be handled, the initial cost of installation is not so

Table III—Drifting, Crosscutting, Raising, and Sinking Data

Kind of Operating	Main haulage	Upper levels	Small drifts and crosscuts	Powder drifts	Ore transfer raises	Stope raises	Prospect winze
Cross-section:							
Height, ft.	10	7	6½	4	7	4½	7
Width, ft.	10	5½	4½	3	7	4½	11
Area, sq.ft.	100	38.5	29.3	12.0	49.0	20.5	77.0
Character of material	Dense gabbro	Dense gabbro	Hard slate	Hard slate	Gabbro	Gabbro	Hard gabbro
Drills	IR N-72	IR N-72	IR N-72	IR N-72	IR R-51	IR R-51	IR R-39 wet
Number per face	2	2	1	1	2	1	3
Round — type	Pyramid cut	Pyramid cut	Pyramid cut	Parallel cut	Pyramid cut	Pyramid cut	V cut
Number of holes	25	18	14	10	16	12	28
Average diameter of hole, in.	1⅞	1⅞	1⅞	1⅞	1⅞	1⅞	1 1¼
Holes per round, ft	175	126	98	70	127	85	150
Average drilled per machine-drill shift, ft.	55	63	53	55	48	48	30
Advance per round, ft.	6.6	6.75	6.0	6.3	6.17	5.9	4.74
Advance per machine-drill shift, ft.	2.1	3.37	3.23	4.9	2.3	3.31	.95
Explosives, strength, and kind	40% Am. L.F.	45% Am. L.F.	45% Am. L.F.	45% Am. L.F.	45% Am. L.F.	45% Am. L.F.	40% gelatin
Size of cartridges, in.	1¼x8	1¼x8	1¼x8	1¼x8	1¼x8	1¼x8	1x8
Explosive per foot advance, lb.	17.90	14.9	9.75	6.40	16.00	9.82	22.0
Crew per shift:							
Drilling	2	2	1	1	2	1	3
Mucking	3	2	1	1	1 nipper	1 nipper	1 hoist man
Total cost, per foot	$20.80	$14.40	$10.95	$8.60	$16.00	$12.00	$33.53

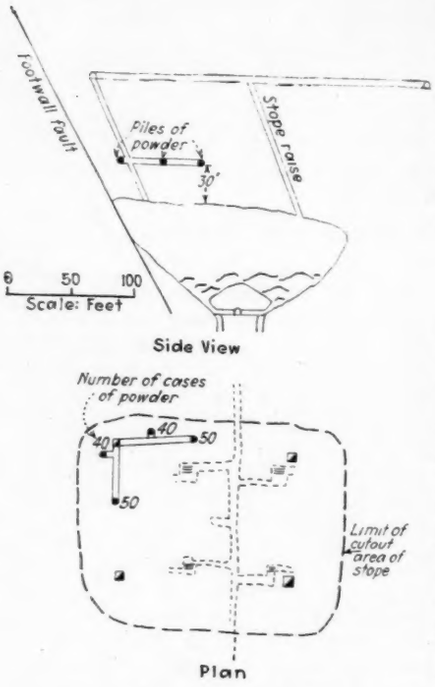


Fig. 6—Typical powder drift shot in the back of a stope

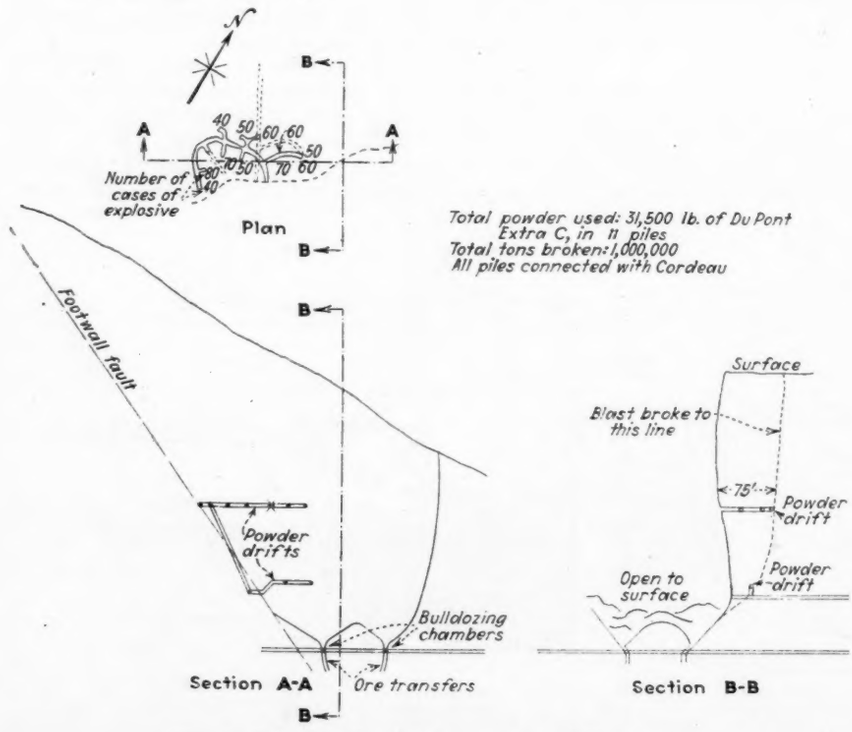


Fig. 7—Powder drift shot in west end of No. 3 cone

important as the durability of the finished structure. The long ore transfers, when first driven, were equipped with a chute-and-timber structure similar to that used for the short chute raises. These served well until the stopes broke through to the surface, permitting entry of surface water and some clay and gravel into the stope opening. Much trouble was then experienced with the chutes. At times the rock, with some mud, would hold back the water until a considerable head had accumulated. When this happened, pressure developed beyond that which the gate and the wooden structure could withstand.

Main loading chutes are now constructed with 15-in. steel girder beams for posts. They are anchored to a concrete foundation, the upper end being secured by a deep hitch in the solid rock. The bottom of the chute is a pad of 6-in. timbers, resting on concrete and lined with 2-in. manganese-steel plates, 30x60 in. in area. Manganese steel is also used for side liners and head-block liners. The air cylinders for the operation of the check gate and arc gate are 6x30 in., and 12x26 in., respectively. Diameter of the piston rods in both cylinders is $2\frac{7}{16}$ in. Cylinders are cast in the machine shop. Four-way throttle valves, $1\frac{1}{4}$ in., are used, and cylinders are connected with flexible metallic hose.

The check gate is made up of cast-steel brackets holding 6 in. shafting. Space between the rail and the bottom of the platform stringer is 7 ft. 2 in., which insures clearance for the load all the way to the tipple house. Good control is obtained for the flow of broken rock with this chute and gates, even when contending with large quantities of water. Design of the chute gate and structure is shown in Fig. 8.

When these heavy gates were developed, raises were driven parallel to the main ore transfer, with openings at frequent intervals to prevent an accumulation of water in the ore transfer. Now all transfers have these parallel raises driven when the raise is driven. To prevent the rock, as well as the water, from passing out through the relief raises, the lower openings are heavily grated; or, in lieu of the grating, several 3-in. holes are drilled through the solid rock to connect relief raise and ore transfer. Details of the water-overflow raise are shown in Fig. 9. Although these gates do not stop the flow of water, mud, and fine gravel, they do control the coarser material, and rarely more than two or three cars of fines are spilled from them at any time; whereas, previously, several cars adjacent to the chute would be completely buried by a spill of material that included coarse ore with the fines.

Timbering is seldom used in raising. Whenever the raise is to be more than 150 ft. long a parallel manway raise is driven, with frequent connections, as shown in Fig. 10. This gives ready access to the work, as the base of work is always from the last connection between the manway and the chute raise, and makes for better ventilation. These raises are generally driven at an angle of between 50 and 70 deg., 55 deg. being considered the most favorable. The timbering of the two prospecting incline shafts now being driven is of 8x8 in. fir-timber sets at 6-ft. centers, with conventional framing. The ground stands well and needs little support.

Operations required in delivering rock from the mine to the concentrator can be easily visualized from Fig. 11. The caved or blasted ore is drawn from the stopes and put through 25-in. grizzly openings by a group of men called "bulldozers." The transfers below the bulldozing chambers average 250 ft. long, generally terminating in a pocket holding from 200 to 2,000 tons of ore. At this point on the main level the ore is drawn through loading chutes into 10-ton cars. At dry chutes the heavy check gate is replaced by a lighter one, and the check-gate cylinders are dispensed with. Hand-operated chutes are still used in some parts of the mine.

When the train of forty 10-ton cars arrives at the main station in the mine it is divided into three sections, and taken over by crews who load at different chutes. This enables 40 cars to be loaded in 15 to 30 min. Each crew consists of a chute puncher, who is the leader; a motorman; and two loaders. By means of a torpedo switch on the loading platform and a string of lights in the drift, signals are flashed to the motorman. Whenever a train must be backed into a loading drift, a system of flashing lights, together with an electric siren, is used. The chute puncher is also required to precede the train. As soon as each section is loaded.

Table IV—Winze Sinking Data

	No. 53 Winze	No. 91 Winze
Formation	Metagabbro with quartz stringers	Slate and gabbro
Character of ground	Hard	Medium hard
Water	Less than 1 g.p.m.	About 1 g.p.m.
Inclination	60 deg.	62 deg.
Sectional dimensions of shaft, ft.	11x7	13x7
Compartments	2	3
Hoisting compartment	4'0"x4'6", in clear	(2) 4'0"x4'6"
Manway	4'0"x4'6"	(1) 2'7½"x4'6"
Distance between sets, ft.	5'11½", c. to c.	5'11½", c. to c.
Size of timber	8x8"; dividers, 6x8"	8x8"; dividers, 6x8"
Kind of timber	Douglas fir	Douglas fir
Capacity of buckets	¾ ton	¾ ton
Depth sunk, ft.	1,100	488
From	July, 1930	Oct., 1931
To	April, 1932	April, 1932
Number of shifts per 24 hr.	Three, 8 hr.	Three, 8 hr.
Sinking crew: — Bottom	9 total, 3 per shift	12 total, 4 per shift
Top	3 hoist men, 1 per shift	3 hoist men, 1 per shift
Wages: — Shaft men	3 contractors, averaging $9.88 per shift; others $5.50	3 contractors, averaging $10.43 per shift; others $5.50
Top men	$4.50	.....
Hoist men	$4.50	$4.50
Kind of rounds	V-cut	V-cut
Kind of drills	R-39 wet	R-39 wet
Number of holes per round	28	34
Depth of rounds, ft.	5	5
Explosives	40% Dupont Special Gelatin	40% Dupont Special Gelatin
Detonators	Dupont California Electric No. 8 caps, 8 relays, 8' leads	Dupont California Electric No. 8 caps, 8 delays, 8' leads
Method of shoveling	Direct into bucket	Direct into bucket
Drill Steel:		
Size, in.	¾ hexagonal hollow	¾ hexagonal hollow
Starter gage	2-in. double taper and bit	2-in. double taper and bit
Gage change, in.	$\frac{3}{16}$	$\frac{3}{16}$
Change in length, in.	18	18
Advance per 8 hr., ft.	0.89	1.10
Advance per machine-drill shift, ft.	0.95	0.68
Advance per round, ft.	4.74	4.10
Sinking costs per ft.:		
Direct labor:		
Shaft men	$19.04	$25.65
Hoist men	4.63	4.29
Top men	2.31	.....
Indirect labor:		
Timber forming	0.68	1.01
Steel sharpening	0.42	0.42
Pipe men and electricians	0.12	0.12
Supervision	1.00	1.00
Supplies		
Timber	1.02	1.70
Hanger bolts	1.19	1.19
Explosives	4.15	5.66
Pipe line	0.11	0.11
Drill repairs	0.25	0.25
Ventilation	0.40	0.40
Miscellaneous	0.25	0.25
Power		
Drilling	0.80	0.85
Hoisting	0.40	0.35
Ventilation	0.05	0.05
Pumping	0.33	.....
Grand total	$37.15	$43.30

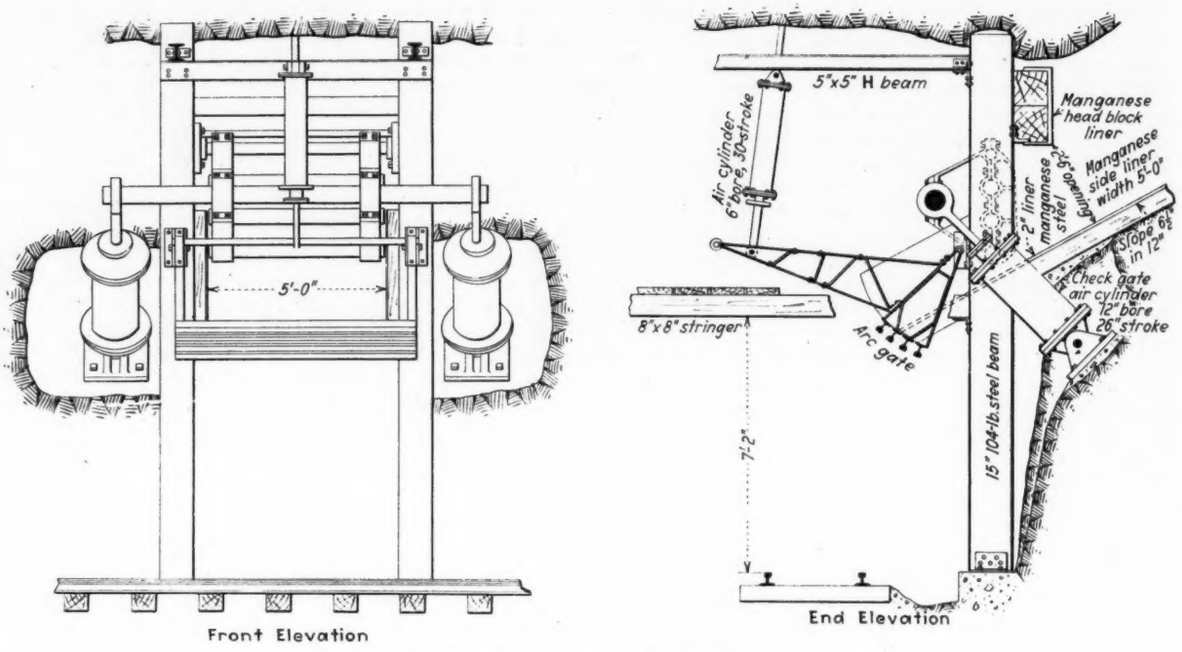


Fig. 8—Air-operated loading chute gate

it is hauled to the main station, where it is made up into a single 40-car train and delivered to the hauling crew, which consists of a motorman and conductor. The train is then hauled about 2 miles to the concentrator on the beach.

The main adit, which is single-tracked, corresponds to half that length of transport. Snowsheds on the side hill, and short tunnels, both double-tracked, cover the remainder of the distance to the tipple house. Three strings, each of 40 cars, and two haulage locomotives are operated. Another haulage locomotive of the same type as the other two is used as a spare. Normally a loaded train meets an empty train on the double track, while the third train is being loaded by the mine loading crew. As with all other mine operations, tramming is done on three shifts.

The ore cars are of the rigid, flat-bottomed, steel box type, having a body 12 ft. 3 in. long, 5 ft. wide, and 3 ft. 8 in. high, with a capacity of 10 tons. The top stands 5 ft. 8 in. above the rail. In all, 186 cars of this type are in commission. Track gage is 30 in. The car is equipped with four cast-iron, chilled-tread wheels, 20 in. diameter. Wheel base is 5 ft. 8 in. Eight single-coil springs of $\frac{13}{16}$-in. material within steel pedestals support the car. Bearings and outside journals of railroad type are used. The bearings are oiled once a month, the year around, with winter non-freezing oil. Wheels are pressed on 4.4-in. diameter axles, which are machined from bar stock in the company's machine shop.

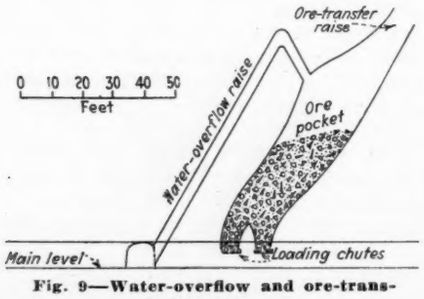


Fig. 9—Water-overflow and ore-transfer raises

To decrease the number of broken axles, carbon steel has been replaced by chrome-vanadium and chrome-manganese steels. When the axle returns to the shop for a new set of wheels it is carefully examined, with a magnifying glass, for flaws. An axle should wear out at least two sets of wheels. Broken axles now occur at the rate of three or four a month, which is not excessive, considering the severe treatment they receive.

The cars, built in the company's shops, are smooth inside, and are strengthened by vertical channels on the outside. Formerly, a triangular brace was used on the inside to keep the side opposite the loading chutes from bulging out. This brace, however, was responsible for the rapid building up of material on the inside of the car, and for this reason it was taken out. No special protection is provided for the bottom of the cars. Each car handles about 30 tons of ore per 8-hr. shift, and is equipped with $\frac{3}{4}$ size automatic, standard drawheads, cast in the company foundry. Draft springs are single-coil, 7 in. long, made of $1\frac{1}{4}$-in. round material. For handling supplies, 25 flat cars, having 12-in. roller-bearing wheels, are in use.

Track gage is 30 in., and the grade 0.5 per cent in favor of the load. Tracks are laid with 50-lb. rails on 6x8-in. fir ties, spaced 2 ft. apart, center to center. The ties used are $4\frac{1}{2}$, 5, and 6 ft. long, depending on the amount of room available. The track is ballasted with minus $\frac{3}{4}$-in. crushed rock, tamped with pneumatic tie tampers. Switches are of the split type, with manganese-steel tips. They are operated from standard railroad ground-throw switch stands. Manganese-steel frogs are used throughout the mine, Nos. 7 and 4 being standards. The sharpest curve is one having a radius of 150 ft., but most of the curves have much greater radii. The rails are electrically bonded with pin-driven copper bonds.

Locomotives used in hauling are equipped with air brakes, and the first four cars in each train are also so equipped. This gives good control. At first, more air-brake cars were used, but the number was reduced, thereby reducing the cost of car upkeep. Trains are dumped in one of two rotary tipples that accommodate four cars at a time. The cars are pushed into the tipple by the haulage locomotive. Signals are flashed by means of torpedo switches, and the proper stopping place for each four-car section is also designated by lights. The crew for dumping consists of the two men in the hauling crew and a tipple man, who remains at the tipple.

At the tipple, every morning, each train of cars is cleaned by the applica-

tion of four air-operated paving breakers on the bottoms, while the tipple is in the inverted position. Formerly, especially in freezing weather, this accumulation proved troublesome.

Telephone communication is provided between main points on the railroad. At the portal of the main adit, where the double track starts, a switchman is posted to regulate traffic on the single track. He also counts the cars, watches for the tail light, and, when a broken train or a broken axle occurs, signals the motorman of the train. Rules for handling trains, to promote safety and to facilitate operations, have been printed in booklet form and are given to each of the operating crew.

A low-density (Du Pont Extra "C") 45 per cent dynamite is used for blasting powder drifts, blockholing, and bulldozing; and 40 per cent Du Pont special gelatin is used in shaft sinking and in some other development headings. The total dynamite used averages 0.34 lb. per ton of ore. Average daily consumption of dynamite, about 80 cases of 50 lb. each, is delivered to the mine from the local magazine every other day. The magazine—convenient to the track that leads to the mine, and having a projecting spur of the mountain between it and Juneau—is bullet- and weather-proof, besides being fireproof, well ventilated, and not heated. Its position is in compliance with the American table of distances. The building, 40x25 ft. in area, shown in cross-section in Fig. 12, is constructed of corrugated iron and weak mortar, in accordance with the recommendation of the Institute of Makers of Explosives. About one month's supply of powder is kept on hand in this local magazine.

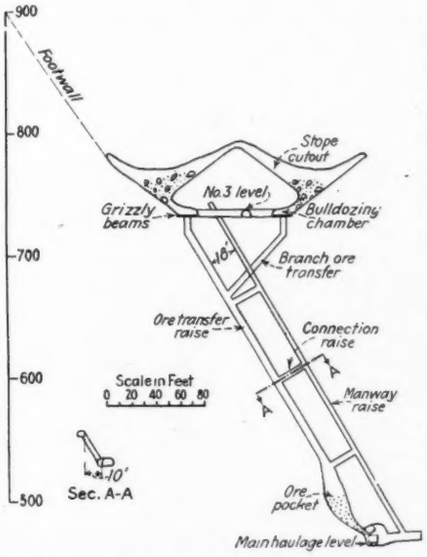


Fig. 10—Ore transfer and connecting manway

Deliveries from the Du Pont company's magazine, on Gastineau Channel, seven miles south of Juneau, are by scow and tug boat. Seven hundred and twenty cases, or ten carloads, is delivered at one time. The cars to be loaded from the scow are lowered by derrick to the deck, and the explosive is loaded directly on the cars. These cars, specially constructed, hold 72 cases each.

In powder-drift blasting the explosive is transported in the cases from the magazine to the end of the nearest track connection. At this point it is emptied into powder bags with a capacity of 50 or 100 lb. each, fitted with pack straps, as at times the dynamite is packed up or down ladders to the powder drift. If it must be moved upward a considerable height, a Tugger hoist is used to elevate the bags up the raise to the powder drifts. Arriving at the powder drifts, the bags are slid on boards and the contents emptied and piled as compactly as possible. The pile is connected to the detonator, and to any adjacent piles that are to be shot simultaneously, with No. 1 wire-countered Cordeau. The explosive is then tamped with the broken muck left in the drift for that purpose. Detonation of the Cordeau is by No. 8 caps and long fuse. Electrical detonation is not employed in blasting powder drifts.

The amount of explosive used in development work is determined by the nature of the ground. All development work is on contract, and the contractor pays for the explosives, so good control is kept of this item. However, all contract work is closely supervised, and the contractors are advised as to the amount of explosive necessary.

As will be evident, the amount of explosive consumed daily produces a large amount of fume. Blasting on the grizzlies and in the development headings takes place whenever the shots are ready. Owing to the generally cool outside temperature, and the arrangement and size of development openings, a strong natural draft is present at all times. This, augmented by several 60-in. fans to help circulation in a few places, gives excellent ventilation. Compressed air is used for brief periods after blasting, to clear the heading and to get the fumes into the main airways. Good ventilation is important, because frequent blasting is essential to the success of the Alaska Juneau method of mining. Powder drifts are blasted only when all the men are out of the mine, and eight hours is allowed to elapse from the time of blasting until work is resumed.

Drilling in all drifts, crosscuts, and stope cutouts is with N-72 Ingersoll-Rand drifters, using 1⅛-in. hexagonal hollow steel. Machines of the anvil-block type are used. Rounds used are shown in Fig. 13. Raising is done with Ingersoll-Rand R-51, self-rotating stopers, using 1⅛-in. hexagonal hollow steel. Blockholing is with Ingersoll-Rand R-12 dry Jackhamers, using ⅞-in. hexagonal hollow steel. Sinking is done with R-39 wet-type Jackhamers, using ⅞-in. hexagonal hollow steel.

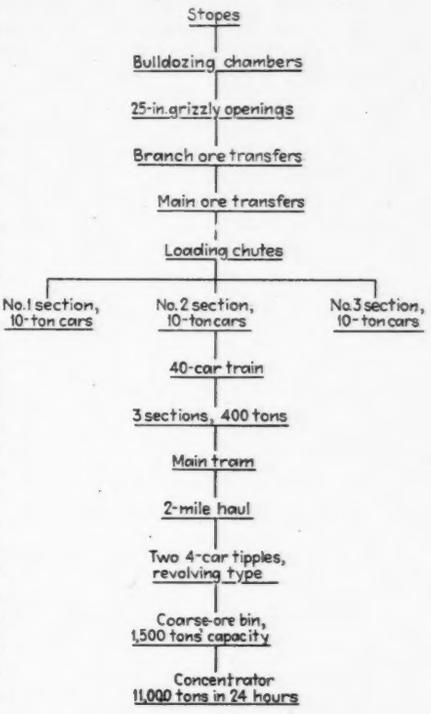


Fig. 11—Ore flow chart

Compressed air is furnished by an Ingersoll-Rand compressor of 4,000 cu.ft. free air capacity, rope-driven from a Westinghouse 750-r.p.m. induction motor. Two additional compressors, one of 750 cu.ft. capacity and the other of 600 cu.ft. capacity, are for emergency use. Compressed air from these three is transmitted through Gold Creek tunnel in two mains, one 5-in. and one 7-in., to a 300-cu.ft. re-

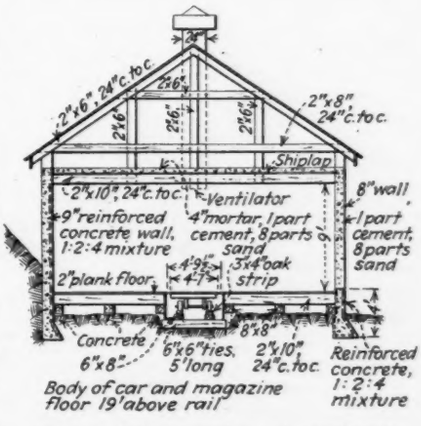


Fig. 12—Cross-section of explosives magazine

ceiver at a point 4,800 ft. from the compressor. Parallel air mains extend from this point through separate drifts on the Gold Creek tunnel level for 3,500 ft., thus insuring an air supply in every part of the mine, even if one air line is out of commission. Several cross-connections are provided between mains, and several branches lead to upper levels.

For use in connection with the deep development program a 2,600-ft. Ingersoll-Rand compressor, direct-connected to a 500-hp. Westinghouse synchronous motor, has recently been installed, with a 10-in. air main to the two prospect shafts. The 4,000-ft. compressor and the 2,600-ft. compressor are each equipped with aftercoolers, and the pipe lines are cross-connected.

Output averages 16½ to 17½ tons per man per day. In obtaining this figure the total of all employees is used, including power plant, foundry, office, and all departments. Bulldozers average 123 tons per man per day; loading and tramming crews handle 163 tons per man per day.

Average development progress in the 5x7 ft. drifts and crosscuts is 3 ft. per machine-drill shift. In the 7x7 ft. raises, progress is 2.0 ft. per machine-drill shift. The smaller raises are driven 2.5 ft. per machine-drill shift. Powder drifts, 3x4 ft., are driven at the rate of 4 ft. per machine-drill shift.

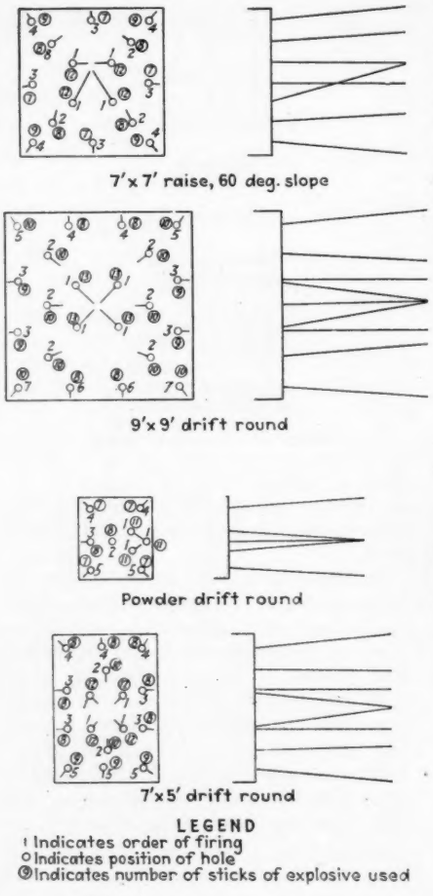
In the north half of the mine, on the No. 4 bottom level, two winzes are being sunk as part of a program for exploring the ore zone below the main adit. The country rock consists of hard slates and metagabbro, with quartz stringers. Winzes are being driven at an inclination of 60 deg., almost coinciding with the dip of the formation. Sinking equipment consists of a ¾-ton self-dumping lugged bucket, sliding on 4x6-in. skids, and a 6½x8-in. air hoist.

Three shifts are worked in each shaft. In one shaft the shift consists of three miners and a hoist man; in the other, of four miners and a hoist man. The miners drill, shovel, or timber, whichever is required. Shoveling is done directly into the ¾-ton sinking bucket. At the top the muck is dumped into a chute in one of the loading drifts and is drawn by the regular loading crews.

The rounds are drilled with hand-held Jackhammers of the wet type, using ⅞-in. hollow hexagonal steel. Change in gage is 1/16 in., size of the finishing drill being 1⅜ in. Change in length is 18 in., the finishing drill being 7 ft. The standard double-taper cross bit is used on all drills.

Rounds consist of 28 to 34 holes. The eight V-cut holes are drilled and blasted. After mucking, the remainder of the round is drilled and blasted. If another set of timber is needed at this point, it is then placed in position, whereupon mucking begins.

Blasting is done with 1x8-in. gelatin dynamite, detonated by No. 8X electric delay blasting caps, the electric det-


7'x7' raise, 60 deg. slope
9'x9' drift round
Powder drift round
7'x5' drift round
LEGEND
! Indicates order of firing
○ Indicates position of hole
⑨ Indicates number of sticks of explosive used

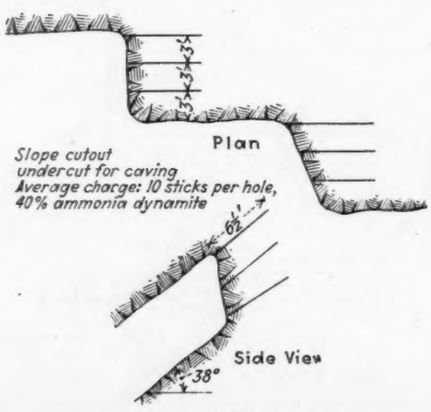
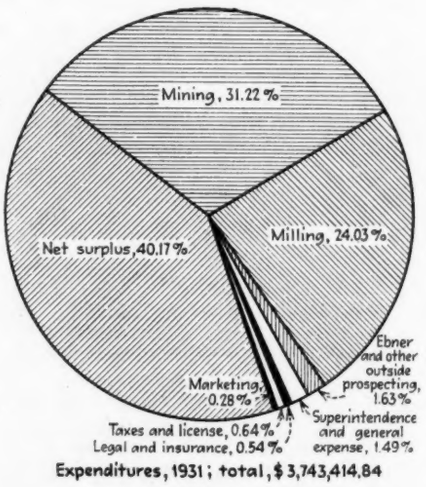
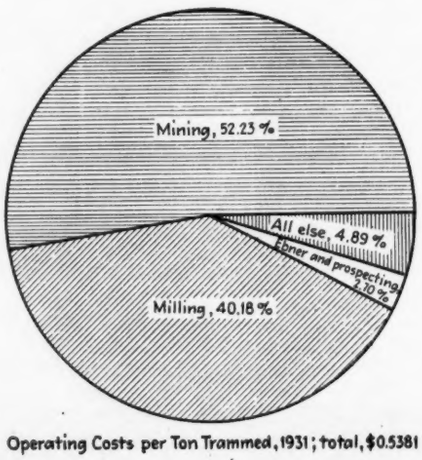


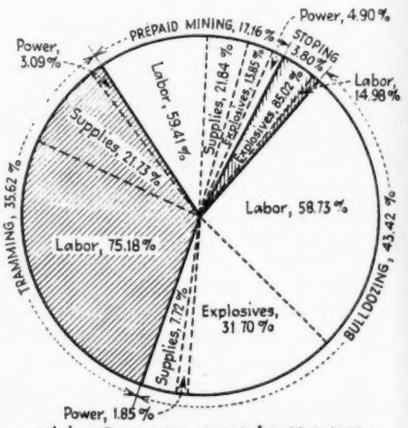
Fig. 13—Drilling and blasting diagrams


Mining, 31.22 %
Milling, 24.03 %
Net surplus, 40.17 %
Ebner and other outside prospecting, 1.63 %
Marketing, 0.28 %
Superintendence and general expense, 1.49 %
Taxes and license, 0.64 %
Legal and insurance, 0.54 %

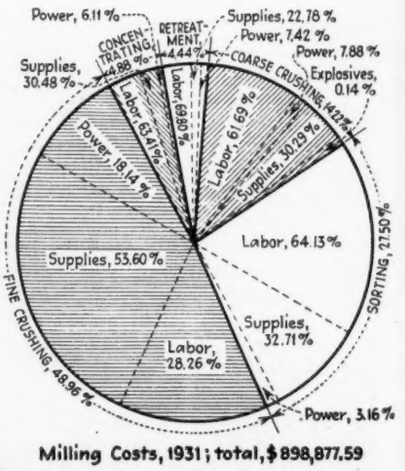
Expenditures, 1931; total, $3,743,414.84



Operating Costs per Ton Trammed, 1931; total, $0.5381



Mining Costs, 1931; total, $1,169,028.45



Milling Costs, 1931; total, $898,877.59

Fig. 14—Distribution of operating costs

onators being arranged in a parallel series circuit and fired by 110-volt current from the lighting circuit. As the work is being done on all three shifts, the small amount of water flowing in is easily handled by hand bailing.

A contract at a certain price per linear foot has been let to three miners, who are required to do the drilling, blasting, shoveling, and timbering. They pay for additional labor, and purchase explosive and detonators from the company, which provides machines, steel, tools, timber, compressed air, and, besides, draws the muck from the bin. It also furnishes the hoist men and does the pipe fitting and electrical work necessary. The work is supervised in the same general manner as other development work, except that a shift boss superintends the blasting operation. Details for both winzes are given in Table IV.

Haulage track is thoroughly ballasted with crushed rock. Mud and water that spill from the cars as they pass over the track make track upkeep difficult. The mud, which builds up around the ends of the ties, causing poor drainage, is cleaned from the track frequently, but occasionally all the old ballast must be removed and replaced with fresh ballast, old ballast being so mixed with mud that it is useless for the purpose.

Mud that leaks out of and overflows from the cars contains fine particles of abrasive quartz. This material causes unusual wear of track, wheels, car couplers, tipples, and all equipment having a sliding or rolling motion. As a consequence, more track renewal and car repairs are necessary than would be required if dry muck were handled. Consequently, tramming costs are somewhat increased.

Operation has been arranged and equipment provided to accord, so far as possible, with variation in seasonal conditions. In the mine and at the crushers provision is made to handle broken rock with a large quantity of water, the amount of which varies widely. With the present equipment the best conditions obtain when a moderate amount of water is coming down through the broken muck, ore transfers, and chutes. When the amount of water is greatly increased, the work is retarded. If the amount of water greatly decreases, the fines in the broken muck become sticky, build up on the bottom of the drawhole, and do not flow to the grizzly freely. This condition retards bulldozing. If a temperature between zero and 20 deg. above zero persists for a long time, freezing of the broken ore in the open stopes takes place, and drawing is retarded. Cold weather has also caused considerable tramming delay.

Water power from the company's hydro-electric plants is used during the summer run-off season. When the weather is dry or cold this source diminishes, and power must be purchased or generated by steam, causing greatly increased expense. A graphic representation showing the surface and underground distribution of operating costs is given in Fig. 14.

Milling Methods and Ore-Treatment Equipment

Beneficiation practice has been developed to meet conditions involving treatment of an unusually low-grade ore and the need to avoid any step in handling that would necessitate an expense incommensurate with the results that might be obtained by its adoption. Maximum efficiency has been reached in the coarse-crushing department, after many changes, the outcome of experience. Sorting and the discard of waste play an important part, suggestive of a phase in the technique of gold-ore treatment that deserves serious attention elsewhere, especially among those who have doubted the ultimate success of the Alaska Juneau experiment. Roll practice in vogue deserves study; and much of interest and value may be learned from an analysis of the trends that have brought the ball mill, as it is being operated today, to the position of importance and irreplaceability it occupies in the flowsheet. The logic and adequacy of concentration, with unusually large cuts of concentrate and middling, with flotation playing a minor rôle, have been amply demonstrated. Amalgamation serves effectively for final recovery of the gold. The entire operation demonstrates the practicability of a simple though effectively coordinated series of operations, the success of which betokens the application of the highest type of metallurgical skill in the beneficiation of a low-grade gold ore.

W. P. Scott
Mill Superintendent

Milling methods and equipment in use at the Alaska Juneau, as in so many other plants, are the result of a gradual evolution in practice. The present concentrating plant, put into operation in April, 1917, was designed to treat 8,000 tons of run-of-mine ore per 24 hr. Plus 6-in. material was to be crushed in two 36x48 in. jaw breakers; the discharge from these and the minus 6-in. material to be crushed by four 9-K gyratories, delivering a minus 3½-in. product to the ball-mill feed bin. This product proved coarser than expected, because of the tendency of the gangue material to break into slabs, some of which were so large that they choked the feed trunnions of the mills. The hardness and toughness of the ore had not been sufficiently realized, with the result that the ball mills did not reach 50 per cent of expected capacity. Hand sorting, employed in some of the early stamp mills, was introduced, and a vigorous campaign of alterations started in an endeavor to attain anticipated plant capacity.

The present coarse-crushing and sorting department consists of three units, each having two 60-in. Stephens-Adamson apron feeders, one 36x48-in. Type "C" Buchanan jaw crusher, set at 8 in.; one 5-ft. bar grizzly with 4-in. spacing; one 42-in. sorting belt; one 9-K Gates-type gyratory crusher, set at about 2 in.; one 3x6-ft. Niagara vibrating screen, with 3-in. square-mesh screens; one 7-ft. bar grizzly, with 1½-in. spacing; and the necessary product conveyors, chutes, and other accessory equipment.

Tramming continues throughout the 24 hours, but the crushing plant is operated for two 8-hr. shifts only, with three hours between shifts, and one hour from each shift for lunch. Four 10-ton mine cars are dumped at a time, by revolving tipple, into the coarse-ore bin, which has a live capacity of 1,500 tons, so little leeway is available for interruption in tramming or crushing. Only two crushing units are operated at one time, the third unit being held in reserve, to allow necessary repair work.

Ore, as delivered to the mill, varies from slime to slabs that will pass a 27-in. grizzly in the mine bulldozing chambers. It has a varying moisture content. Most of the ore cars arrive at the mill partly full of water, and this water, with the fine granular part of the material, causes excessive wear on the feeders and on all crushing equipment. No screening is practiced ahead of the crushers. Occasional large slabs block the feeders or hang up in the crushers,

and have to be blasted in place. The life of the feeders has been almost trebled by installing manganese-steel bushings and links, lasting six months and one year, respectively. Manganese track shoes last about five months. One trouble—escape of wheels off feeder shafts—has been minimized by a change in the method of locking the bushing in place. Manganese wearing bars are used on the feeder plates.

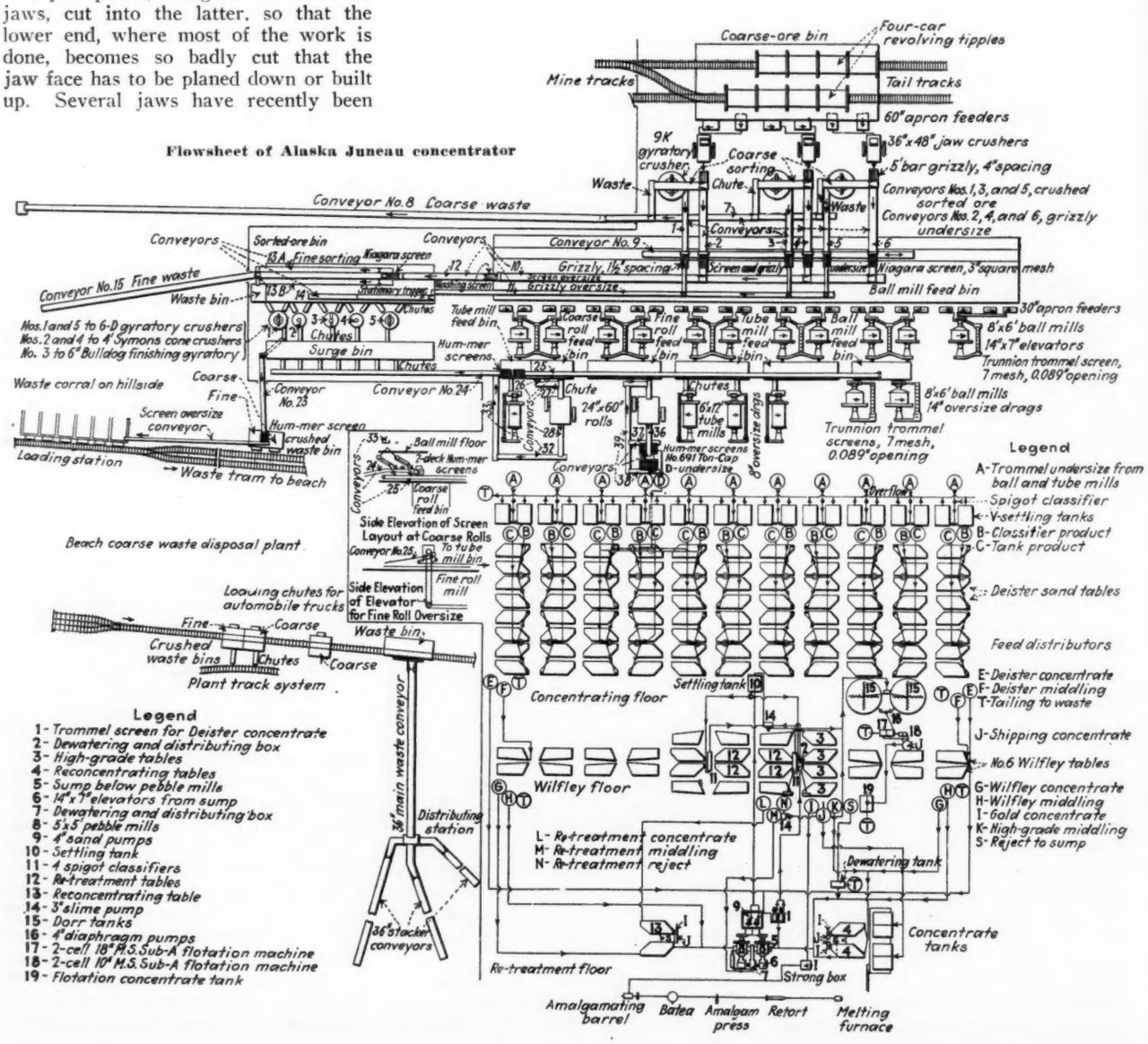
The jaw crushers are operated at 166 r.p.m., and each belt is driven by a 150-hp. squirrel-cage induction motor, direct-connected to the line shaft. One unit is driven by the original installation, which consists of a 350-hp. synchronous motor, direct-connected to a line shaft, equipped with Allis-Chalmers friction clutches, driving both the crusher and gyratory. The manganese steel jaw plates, with corrugated faces, are 4 in. thick. These plates, used in four positions to get maximum service, last about six months. The jaw plates, being harder than the jaws, cut into the latter, so that the lower end, where most of the work is done, becomes so badly cut that the jaw face has to be planed down or built up. Several jaws have recently been built up and faced with a hard facing material that resists the action of the jaw plates.

In installing the sorting plant a grizzly had to be placed immediately under the discharge of the jaw. This often causes long slabs to hang up against the bottom of the jaw plate, and results in excessive wear on the toe lugs, on some occasions breaking off the end of the jaw casting that holds these lugs. To avoid this trouble, lugs have been cast on the back of the jaw plates, and fitted into slots cut into the face of the jaw, a wedge key at the top holding the jaw plate locked in place. These have proved satisfactory, making serviceable old jaws, the discarding of which otherwise would have been necessary. Old jaw plates are cut up and used for waste chute liners.

Sorting is done in two places. Coarse sorting with the original arrangement of equipment involved working within a limited space. A 5-ft. grizzly, with tapered bars spaced 4 in. apart, is placed directly under the jaw and as close to the discharge as practicable, to remove water and fine material. The oversize discharges on the end of a 42-in. sorting belt, traveling 100 ft. per minute, placed directly over the bowl of the 9-K gyratory crusher so that the ore pulled off the belt falls directly into it. A heavy spray of water washes the material as it comes onto the belt. Four sorters work at each belt and pull off all the ore, 70 to 80 tons per man per 8-hr. shift, leaving the waste on the belt, for passage through a chute to conveyors, which deliver it to a waste bin on the hillside.

Alaska Juneau ore is well suited for hand sorting. Almost all the valuable minerals occur in the quartz, or at the contact of the quartz and the gabbro. No special skill is required in sorting, which involves selecting the white

Flowsheet of Alaska Juneau concentrator



quartz, or pieces with adhering quartz, from the darker gangue material. The metagabbro, the principal gangue material, is extremely hard and tough, and difficult to crush. It tends to break into slabs, so that large slabs of waste are pulled from the belt with the ore, to avoid hang-ups in chutes and conveyors. The company furnishes all the sorters with cotton gloves; also leather palms for the coarse sorters and rubber finger cots for the fine sorters. Frames of the conveyors have been made as narrow as possible, and idler bearings of special design are placed under the belt, insuring minimum resistance to the pulling of this coarse material over the side of the belt. Sorting belts have steel frames and can be lifted out by the crane for repairs or for installation of new ones. Rubber-covered idlers are used at loading points on conveyors. A folding steel platform over the belt protects the pickers from falling objects.

Percentage of oversize on the 4-in. grizzlies ranges between wide limits, depending on mine and weather conditions. During rainy weather an excess of fine material is washed down through the open stopes; and in cold, dry weather the proportion of coarse material increases. An average of 40 to 45 per cent of the material trammed from the mine passes over the coarse sorting belts; and 1,000 to 1,200 tons per 24 hr. is sorted out as ore. This material is crushed in the gyratories, and originally went to the ball-mill feed bin. On account of the tendency of the rock to break into slabs, the crushers had to be kept set so close that excessive wear developed on parts, and breakage of cone shafts occurred. A conveyor was installed to carry this material, after it had passed over a fine grizzly, to a sorted-ore bin, to keep it out of the ball-mill feed. This allowed the gyratories to be opened up, and upkeep has been materially improved thereby. Extra thick concaves have been installed; and worn mantles, built up with the electric arc, are giving satisfactory service. The gyratory, operated at 300 r.p.m., is driven by a 150-hp. motor with belt drive.

Undersize of the 4-in. grizzly passes through a chute with cast-steel screen bottom, to remove excess water, otherwise at times this material would not stay on the 42-in. conveyor, which carries it to the Niagara screen. The sieve material is of cast manganese steel, made in two sections, so that both can be changed end for end, and reversed for additional service. The cast manganese steel has proved cheaper than any other screen tried. All woven screens have to be spot-welded at each intersection of the rods; otherwise they soon wear out. Niagara screens, although working under severe conditions, have given excellent service, and have effected a material reduction in screening costs. The expense of operation and maintenance is 0.1 to 0.2c. per ton handled. Undersize of these screens is distributed by a 36-in. shuttle conveyor into the ball-mill feed bin, which has a live capacity of about 5,000 tons. An average of 4,500 to 4,800 tons per 24 hr. is now fed into this bin.

Oversize of the Niagaras is conveyed to a stationary washing screen, then elevated by an inclined conveyor to a second Niagara having the same size and type of screen as the first. The purpose of this screen is to remove any undersize that passes over the first screens as a result of fluctuations in the rate of feed, which varies from 2 tons per minute to short surges of 8 to 10 tons per minute. Undersize from this screen drops into the sorted-ore bin, and the oversize falls on two 42-in. sorting conveyors, traveling 150 ft. per minute, above the top of the sorted-ore bin. Four sorters at each belt, as in the coarse sorting, pick off the ore at the rate of 20 to 25 tons per man per 8 hr., dropping it directly into the bin below, and leaving the waste on the belt. This fine waste is conveyed to the waste bin on the hillside or crushed in a Symons cone crusher, screened, and conveyed to a storage bin, which supplies crushed rock for concrete and road-surfacing material. Ore sorted and screened at this point averages 500 to 700 tons per 24 hr.

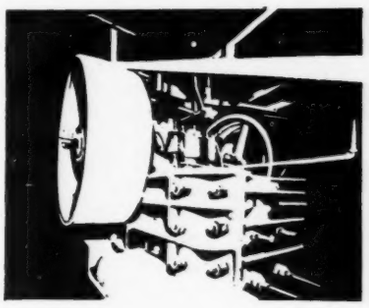
An outstanding feature of present operations is the discarding, by hand sorting at low cost, of a large percentage of the material trammed from the mine, leaving a product of sufficient value to stand the cost of fine grinding and necessary treatment. Sorting also gives the mill a capacity of 80 to 120 per cent more than the fine-grinding capacity. During part of 1927 a daily average tonnage of about 13,000 tons was handled, 57 per cent of this being rejected as waste. At present about 11,000 tons is handled per 24 hr., with a little over 40 per cent rejected as waste. Many features of the present sorting and crushing layout are not ideal, but the sorting plant has been worked into a crowded installation, where no provision had been made for it, without interruption of operations. The progress made in increasing capacity and reducing costs, since the plant went into operation, is shown in Table I.

Table I—Increase of Mill Capacity and Cost Decrease

Year	Number of Days in Operation	Tons Mined per Day	Tons Fine-Milled per Day	Tons Coarse Waste Rejected per Day	Cost of Milling per Ton Trammed	Gold Assay Value	Gold Recovered
1917	274	2,472	2,472	....	$0.7171	$0.86	$0.64
1918	363	1,631	1,582	49	0.6762	0.92	0.73
1919	363	1,909	1,698	211	0.5608	0.99	0.72
1920	364	2,590	1,751	839	0.4463	1.08	0.78
1921	363	4,445	2,491	1,954	0.3554	0.86	0.60
1922	361	6,400	3,070	3,330	0.2613	0.81	0.56
1923	361	6,860	3,144	3,716	0.2662	0.81	0.58
1924	362	8,476	3,778	4,698	0.2512	0.87	0.62
1925	362	9,618	4,248	5,370	0.2504	0.85	0.58
1926	362	10,579	4,557	6,022	0.2154	0.75	0.50
1927	362	11,790	5,083	6,707	0.2257	0.77	0.55
1928	363	10,243	4,946	5,297	0.2285	1.11	0.84
1929	362	10,598	5,581	5,017	0.2319	1.12	0.89
1930	362	10,841	5,708	5,133	0.2271	1.10	0.86
1931	362	11,498	6,351	5,147	0.2160	1.12	0.89

Percentage of coarse waste rejected since 1927 has decreased, because of change in the character of ore delivered to the mill.

Ore from the sorted-ore bin was first crushed in 6-D Gates gyratory crushers and conveyed by launder to the tube mills. This was not satisfactory, so feed bins for the mills were installed, and the material was sent to them. When decision was made to remove the 9-K gyratory product from the ball-mill feed bin, and put it into the sorted-ore bin, additional crushing capacity had to be provided. Two 6-in. Bulldog finishing crushers were installed to aid in crushing the material from the sorted-ore bin, and three of the tube mills were taken out to make room for the installation of two sets of Traylor AAA, Ajo-type, 24x60-in. rolls. The 6-in. Bulldog finishing crusher had too small a feed opening for the size of material handled, and required constant attention to avoid bridging over with slabs and interruption to feed. Two 4-ft. Symons cone crushers were installed and are now used exclusively, the others being held in reserve for emergency. These crushers are giving excellent service and have a capacity of about 125 tons per hour when set at 1 in. Considerable trouble was experienced at first be-



cause of wear of the oil pumps, but as soon as the new type of dust seal and air manifold was installed, this ceased. In cold weather the oil is kept warm by a bayonet-type electric heater in the oil tank. These crushers are driven by 100-hp. ball-bearing induction motors, with Texrope drives. Typical screen analyses of the feed and product of the cone crushers are given in Table II. The plus-1-in. material in the product consists of slabs one inch or less thick.

Cone-crusher product is conveyed, without screening, to the coarse-roll feed bin. Short belt conveyors, one of which is equipped with a Cutler-Hammer magnetic head pulley, feed this material to the rolls. When the rolls were first installed the claim was made that they could take a minus 1½-in. feed and reduce it to minus 7 mesh at the rate of 50 tons per hour, but the ore was so hard and tough that the machine was nearly shaken to pieces, and the tonnage could not be handled. Decision was then made to use one roll as a coarse unit and the second one for fine crushing. The product of the coarse roll is elevated by a series of inclined conveyors to the level of the ball-mill floor, where it is fed onto two double-deck Hum-mer screens. The upper screen has ⅞-in. square openings, and the lower one is a Tyler No. 691 Ton-Cap. Oversize of the ⅞-in. screen falls onto the same conveyor belt that carries the Symons cone-crusher product and returns with it to the feed bin. The product on the fine screen is conveyed to the fine-roll feed bin, and the undersize is conveyed by launder to the concentrating floor. To operate this equipment in the limited space on the ball-mill motor floor, two conveyors had to be installed, one above the other, underneath the two Hum-mer screens, as shown in the flowsheet. An average of about 90 tons per hour is put through the ⅞-in. screen, and 8 to 10 tons of this passes through the fine screen. The circulating load is about 100 per cent.

The fine rolls are fed in the same manner as the coarse rolls, except that no magnetic head pulley is used. The product of these rolls falls directly onto an incline conveyor, which delivers it to four 4x5-ft. single-deck Hum-mer screens equipped with No. 691 Ton-Cap screens. Undersize goes to the concentrating floor, and oversize is conveyed back to the level of the roll feeder, where it goes back onto the feed conveyor or into a 14x7-in. bucket elevator, to be discharged to a conveyor above the tube-mill feed bins, where it is distributed by stationary trippers. An exhaust fan at each set of rolls and each of the four Hum-mer screens removes the dust to the outside of the mill building. An average of about 50 tons per hour is finished on these rolls, the circulating load being from 150 to 200 per cent. Typical screen analyses of feed and finished product on the fine rolls are given in Table III.

Both sets of rolls were originally driven at 100 r.p.m., but the speed of the fine rolls was cut to 80 r.p.m., with marked increase in tonnage crushed. Each set of rolls is driven by two 150-hp. induction motors, with 20-in. 8-ply endless belts and idlers. Bearings are babbitted undersize, then turned to fit in a lathe. This gives a bearing the breaking in of which causes no trouble. The original bearings gave considerable trouble by heating, especially those on the fine rolls when crowded, so water-cooled bearings have been tried and found much more satisfactory. Felt dust seals give a marked increase in the life of the bearing. Oil feed through drip cups has been found to provide satisfactory lubrication for an ore crusher. Roll shells are cast high-carbon, nickel-chrome steel, 8 in. thick. Shells are placed on the cores cold, pulled in place as tightly as possible, then heated up slowly for about two hours by a kerosene-fired burner, which goes entirely around the shell. They are then pulled on far enough further to allow for shrinkage of 0.018 to 0.020 in. Distribution of the feed into the rolls and lateral adjustment are carefully watched, so that no grinding of the shell faces occurs, and only occasional burning off of the edges is required. The average life of a set of shells on both the coarse and fine rolls is about 2,000 crushing hours, or a total steel consumption of 0.22 lb. per ton.

The mill is divided into eleven sections, each of which is equipped with an 8x6-ft. ball mill, operating at 22 r.p.m. and driven by a 225-hp. induction motor, with 30-in. 8-ply endless belt and idler. The mills are lined throughout with cast manganese-steel liners and grates, and carry a load of about 20,000 lb. of 6-in. forged chrome-steel balls. Each mill is served by two 30-in. Stephens - Adamson apron feeders equipped with revolution counters for calculating the tonnage handled. The ore and some of the water and fine material screened out at the crusher are delivered through a chute to the scoop box of the mills. Excess crusher water is screened in a special trommel with 7-mesh screen, the undersize going to the concentrating floor and the oversize into a tube-mill feed bin. A dilute pulp, about 60 per cent moisture, and 1-in. slot grates insure quick discharge from the mill.

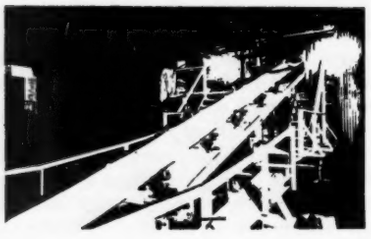
A double trommel screen is bolted to a specially designed flange on the discharge trunnion liner. The inner screen is of ⅜-in. boiler plate with ⅞-in. round holes, and the outer one of 7-mesh screen with 0.089-in. square openings. The purpose of the inner screen is to carry the weight of the whole screen, keep the coarse rock off the surface of the fine screen, and to act as a feed distributor. Undersize of the fine screen is conveyed by launder to the concentrating floor. Oversize of one screen or of both screens is returned to the scoop box of the mill by a 14x7-in. bucket elevator, or dropped by gravity into the feed bin of the tube mill on the floor below. Average tonnage finished through the 7-mesh screen for each mill is 15 tons per hour of grinding time. Average circulating load, when grinding in

Table II—Screen Analyses of Cone-Crusher Feed and Product

Feed			Product		
Size	Per Cent	Cumulative, Per Cent	Size	Per Cent	Cumulative, Per Cent
Plus 5 in.	7.4	7.4	Plus 2 in.	13.8	13.8
Plus 4 in.	19.1	26.5	Plus 1½ in.	23.7	37.5
Plus 3 in.	30.8	57.3	Plus 1 in.	28.6	66.1
Plus 2½ in.	17.1	74.4	Plus ½ in.	14.5	80.6
Plus 2 in.	10.9	85.3	Plus ¼ in.	3.9	84.5
Plus 1½ in.	7.1	92.4	Plus ½ in.	3.4	87.9
Plus 1 in.	7.6	100.0	Minus ½ in.	12.1	100.0

Table III—Screen Analyses of Fine-Roll Feed and Product

Feed			Finished Product		
Size	Per Cent	Cumulative Per Cent	Size	Per Cent	Cumulative Per Cent
Plus ¼ in.	20	20	Plus 10 mesh	2	2
Plus ⅜ in.	25	45	Plus 14 mesh	11	13
Plus 3 mesh	16	61	Plus 28 mesh	26	39
Plus 4 mesh	12	73	Plus 35 mesh	11	50
Plus 7 mesh	14	87	Plus 48 mesh	6	56
Minus 7 mesh	13	100	Plus 65 mesh	5	61
			Plus 100 mesh	6	67
			Plus 150 mesh	4	71
			Minus 150 mesh	29	100



closed circuit, is 135 to 150 per cent. Typical screen analyses of feed and finished product of these mills is given in Table IV.

The original shells of these mills, of 1-in. plate, riveted to cast flanges, gave much trouble because the rivets worked loose and allowed grit and water to get into the gear and pinion, causing excessive wear. Cast-steel shells, installed in all the mills, are a decided improvement over the other type. The average life of a set of manganese-steel grates and feed-end liners is about 5,000 grinding hours; that of the breast liners, about 6,000 hours. Steel consumption, based on tons screened through the 7-mesh screens, is given in Table V.

Originally, one 6x12-ft. tube mill was provided in each section of the mill. Only four of these mills are now in use, the others having been taken out to make room for the rolls and for two 8x6-ft. ball mills. The four mills in use, converted to function as ball mills, are operated at 25 r.p.m. and driven by a 150-hp. induction motor with 24-in. 8-ply endless belt and idler. The 150-hp. motors used on the rolls and at the crushing plant are old tube-mill motors.

Table IV—Screen Analyses of Ball-Mill Feed and Product

	Feed		Finished Product	
	Per Cent	Cumulative, Per Cent	Per Cent	Cumulative, Per Cent
Plus 2½ in.	1.8	1.8		
Plus 2 in.	10.4	12.2		
Plus 1½ in.	19.8	32.0		
Plus 1 in.	24.8	56.8		
Plus ½ in.	8.7	65.5		
Plus ¼ in.	9.6	75.1		
Plus 3 mesh	10.2	85.3		
Plus 7 mesh	9.1	94.4		
Minus 7 mesh	5.6	100.0		
Plus 10 mesh			4.3	4.3
Plus 14 mesh			9.5	13.8
Plus 20 mesh			10.7	24.5
Plus 28 mesh			10.0	34.5
Plus 35 mesh			9.3	43.8
Plus 48 mesh			7.0	50.8
Plus 65 mesh			5.1	55.9
Plus 100 mesh			7.0	62.9
Plus 150 mesh			5.6	68.5
Plus 200 mesh			4.0	72.5
Minus 200 mesh			27.5	100.0

Table V—Ball-Mill Steel Consumption

	Pounds per Ton		
	1929	1930	1931
Manganese-steel breast liners	0.1985	0.2140	0.2350
Manganese-steel feed-end liners	0.0558	0.0588	0.0660
Manganese-steel grates	0.0368	0.0357	0.0300
Total liner consumption	0.2911	0.3085	0.3310
6-in. chrome-steel balls	1.014	1.036	1.119

Cast manganese-steel breast liners are used; but the grates and feed-end liners are of cast steel, made locally in the company's own foundry. Ball load is 20,000 lb., made up of worn balls from the ball mills and some new 5-in. balls. General practice when a ball mill is stripped for relining is to send part of the old ball load to the tube-mill floor. Balls are fed to all mills daily in proportion to the running time of each mill. Material handled by these mills consists of fine-roll oversize, ball-mill oversize, or a combination of the two; it is fed into the scoop box of the mill by a gravity chute, so arranged that the material feeds in as fast as it is taken out by the scoop. A dilute pulp, about 60 per cent moisture, carries the material rapidly through the mill. The grates have ½-in. slot openings. Trunnion trommel screens, similar in design to those in the ball mills, are used. Undersize of the fine screen joins the product of the ball mill, and is conveyed by launder to the concentrating floor. Oversize is returned to the scoop box by an 8-in. belt drag. Circulating load of 75 to 100 per cent is maintained. An average of about 11 tons per hour is finished through the 7-mesh screens. A typical screen analysis of feed and product is given in Table VI.

Average life of manganese-steel breast liners is 2,700 grinding hours; of cast-steel grates, 1,700 hours; and of cast-steel feed-end liners, 1,800 hours. Consumption of steel in pounds per ton finished is: manganese-steel breast liners, 0.856; local cast-steel feed-end liners, 0.060; local cast-steel grates, 0.091; total liner consumption, 1.007; ball consumption, 1.354.

Recently two 8x6-ft. ball mills, similar to the ones on the ball-mill floor, have been installed to replace some of the tube mills. These two mills are each driven by a 225-hp., 327-r.p.m. sychronous motor, direct-connected to the pinion shaft with flexible coupling. Helical gears and pinions are used. The mills, which handle the same kind of material as do the tube mills, and are in closed

Table VI—Screen Analyses "Tube-Mill" Feed and Product

	Feed		Product	
Size	Per Cent	Cumulative Per Cent	Per Cent	Cumulative, Per Cent
Plus ¾ in.	8.0	8.0		
Plus ½ in.	5.5	13.5		
Plus ¼ in.	7.8	21.3		
Plus ¼ in.	10.9	32.2		
Plus ⅛ in.	36.6	68.8		
Plus 7 mesh	1.9	70.7		
Plus 14 mesh	20.7	91.4		
Minus 14 mesh	8.6	100.0		
Plus 10 mesh			3.8	3.8
Plus 14 mesh			8.7	12.5
Plus 28 mesh			19.8	32.3
Plus 35 mesh			11.7	44.0
Plus 48 mesh			6.9	50.9
Plus 65 mesh			5.7	56.6
Plus 100 mesh			10.5	67.1
Plus 150 mesh			2.7	69.8
Minus 150 mesh			30.2	100.0

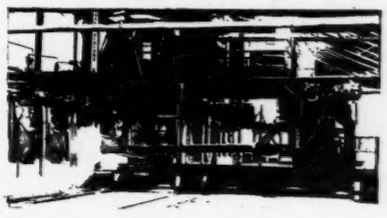
circuit with 14-in. belt drags, have not been in service long enough to permit publication of comparative data.

Distribution of material to the different departments and machines has been made as flexible as possible, to take care of the fluctuations in character of mill feed and to avoid loss of tonnage because of repair work on any particular machine. Spare parts, and in some cases spare machines, are kept ready to install in all important operations. Conveyor pulleys, gears, and pinions have been standardized as much as possible, to insure interchangeability of spare parts. This has materially reduced lost time and lessened the stock of spare parts carried.

Gold, the principal economic metal, is mostly native and usually associated with small quantities of galena and sphalerite. The principal sulphide, by volume, is pyrrhotite, with some pyrite. The gold is coarse, an occasional particle weighing more than a gram. Comparatively coarse grinding liberates the gold. This is of decided advantage in reducing grinding costs, but it yields a product that causes excessive wear of launders and junction boxes. Because of the steepness of the hillside on which the mill is built, a great deal of fall is inevitable in some of the launders, all of which have cast-steel liners, made locally. Junction boxes, lined with concrete and specially designed cast-steel plates, are sealed with lead, to obviate theft. Once each month the boxes are opened and washed out and any necessary repair work is done. Quick-setting cement is used for this work.

Originally, the finished product from the ball mills was conveyed by launder to a rougher floor and passed over Deister simplex sand tables, which made a lead and an iron concentrate. The reject was classified, ground in the tube mills, and sent to the same type of tables on the finisher floor, where a lead and an iron concentrate, a middling, and a waste tailing were made. Roughing tables were discarded, and a machine shop was installed on that floor. Now the first and principal table operation is on what is locally called the finishing floor. Each of the eleven sections on this floor has one classifier, two V-tanks, two feed distributors, and eight Deister simplex sand tables.

Undersize of the 7-mesh screens, from all sources in each section, passes through the classifier—a one-spigot



Deister cone remodeled to suit local conditions and placed in the bottom of the 12-in. launder leading to the V-tanks. This classifier makes a rough separation of the coarser material; its discharge is conveyed by launder to a distributor, which feeds the first four Deister tables in each section. Overflow passes on to the V-tank, where it is thickened. Overflow of the V-tank goes to the tailing flume, and the thickened product to the second distributor, which feeds the last four Deister tables of the section. The original mechanical-feed distributors have been replaced by ones of local design, which consist essentially of two concentric cast-iron pots. The pulp drops through a pipe into the center of the inner and shallower pot, and overflows into the outer one, which has a series of holes uniformly spaced to feed the four tables. These distributors give fairly uniform distribution and require little attention or upkeep, in addition to conserving headroom needed to give the launders a steeper grade. A drop of 2 in. per foot is necessary for launders handling the coarser products. The Deister tables, as originally installed, had the old open-type head motions and "drumstick" under-construction. These proved unsatisfactory and were replaced with Deister Plat-O self-oiling head motion and slipper-bearing under-construction, which have proved satisfactory. Breakage of rocker arms, curved toggle arms, and springs has been the greatest source of trouble.

Covering for the deck, under the riffles, is $\frac{1}{16}$-in. rubber sheeting, with one-ply fabric in the center; linoleum is used for the apron. The rubber lasts several times as long as did the linoleum, especially with the coarse feed. Oak riffles, 1 in. wide and $\frac{3}{8}$ in. high, made in two sections, are spaced 1 in. apart. The 32-in. tips outlast four sets of those at the feed end, the latter serving on an average about three months. Three 35-hp. induction motors drive the tables on this floor and some of those on the floor below. They were originally arranged to drive 40 tables apiece. Tables are operated at 267 r.p.m., with a stroke of $\frac{3}{4}$ to $\frac{7}{8}$ in.

Three products are made by each of the tables on the primary concentrating floor—a rough concentrate, a dirty middling, and a waste tailing. Their primary function is to produce clean tailing without regard to the grade of the concentrate or middling—the reverse of ordinary table work. Middling is taken from between two definite points on the table, regardless of the character of the material that may lie between those points; concentrate constitutes all the remaining material produced, regardless of appearance or condition. Virtually the same amount of concentrate and middling is cut, whether the tables are at normal maximum or at normal minimum loads. Each Deister table produces about 0.75 ton of concentrate and 6 tons of middling per 24 hr. Concentrate consists of nearly all the freed gold and sulphides, plus coarse sand and true middlings. Middling consists of a small amount of freed sulphides and a large amount of pulp containing some true middling; this material is conveyed in launders directly to the Wilfley floor, and becomes the feed for two No. 6 Wilfley tables in each section. The tables recover not only the freed sulphides in this material but also the relatively high-grade coarse sand in quantity sufficient to supplement the pulp in the re-treatment circuit up to the capacity limit of the fine-grinding equipment in that department, and to produce a waste tailing. A minute cut of the freed sulphides, made on each table, is conveyed in launders to the reconcentrating tables on the re-treatment floor; the middling cut, about 1 ton from each table per 24 hr., goes to the sump.

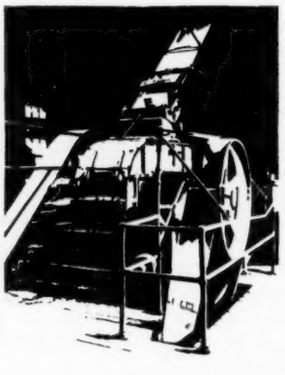
Rough concentrate from all the Deister tables is conveyed in launders to one of two small trommel screens on the re-treatment floor, which separate tramp oversize and send it to the sump. Undersize of the screen is elevated by one of two 2-in. Krogh centrifugal sand pumps to a small V-tank. Thickened product of this tank is fed to four Deister reconcentrating tables, on the Wilfley floor, which produce the first finished concentrate. Two concentrates are made: one, a high-grade gold cut containing as much of the freed gold as possible, is piped to a strong box; the other, a shipping concentrate, is run in launders to the concentrate tanks. A middling cut, consisting mostly of freed sulphides, pyrrhotite, and pyrite with some galena and freed gold, is sent to two Deister reconcentrating tables on the re-treatment floor; the reject is sent to the sump. The purpose of retabling the middling a second time is to remove as much freed gold and galena as possible from this heavy material before it is reground. These tables make a small amount of high-grade gold concentrate, a shipping concentrate, and a reject tailing product that is sent to the sump.

Material from the sump is the feed for the re-treatment department. It is elevated by one of two 7x14-in. bucket elevators to a dewatering box, which feeds the partly thickened material to one or two 5x5-ft. Allis-Chalmers ball mills equipped with shovel-wheel feeders. Overflow of the dewatering box and shovel wheel, together with the product from the mill, is pumped by one of two 4-in. Krogh centrifugal sand pumps to a V-tank on the concentrating floor. The partly thickened product from this tank is conveyed in launders to two four-spigot Deister cone classifiers. Product from each spigot is tabled on one of eight Deister re-treatment tables. These tables recover (1) as much freed gold and galena as possible from this material in a low-grade concentrate, (2) middling which is returned to the sump, and (3) a low-grade reject that formerly went to waste but is now sent to the flotation circuit. Tables handling the product from the first two spigots of each classifier now send all their reject to the sump for regrinding. Low-grade concentrate from these re-treatment tables is conveyed by launder to a Deister reconcentrating table, on the re-treatment floor, which makes a high-grade gold concentrate, a shipping concentrate, and a reject that is sent to the sump.

Flotation plays a minor part in mill operation, treating from 90 to 100 tons per 24 hr. of V-tank and classifier overflows and re-treatment table tailing. These products are delivered by two 3-in. Byron-Jackson slime pumps to two Dorr tanks, 17 ft. diameter, and cut down to 3 ft. in depth, to fit space available. Each is served by a 4-in. Gould diaphragm pump, which delivers the thickened product, about 25 per cent solids, to a launder feeding the flotation machine. A two-cell, 18-in., Mineral Separation sub-A machine is used as a rougher, and a two-cell 10-in. sub-A machine of the same make is used as a cleaner. Each is driven with one motor and V-belts.

The chemicals employed—0.035 lb. potassium amyl xanthate and 0.10 lb. cresylic acid—are fed by disk-and-cup feeders to the launder delivering to the rougher cells. Concentrate from both these cells is fed to the cleaner cells, and tailing sent to waste. Concentrate from both cleaner cells is sent to a settling tank, and tailing fed back into No. 1 rougher cell. A light froth has to be carried to keep the iron down and to make a concentrate of sufficient value to stand freight and treatment charges. Air is furnished by a Roots roller-bearing Acme No. 3 Blower at $1\frac{1}{2}$ lb. Ratio of concentration in this operation is about 100 to 1.

The foregoing description brings out



the point that the table work in the Alaska Juneau mill is different from that in the ordinary concentration mill. Concentrate and middling cuts are both enormously large in proportion to the mineral content in the feed, which is extremely small; an average of 1.15 lb. of lead is recovered per ton of ore finely ground. Average ratio of concentration is about 750 to 1. All tables making a clean waste tailing do not make a finished concentrate, and those making a finished concentrate do not make a waste tailing.

Numerous experiments have been made with finer grinding and better classification of table feed, regrinding of certain products, desliming of table feed and flotation of product, flotation of tailing and other modifications, but the additional recovery has not been found to justify increased cost of operation. A typical distribution of gold in the Deister and Wilfley table tailings is given in Table VII.

Table VII—Distribution of Gold Per Ton of Screened Table Products

Size	Deister	Wilfley
Plus 28 mesh	$0.2766	$0.3063
Plus 35 mesh	0.2747	0.3370
Plus 48 mesh	0.2734	0.3504
Plus 65 mesh	0.2575	0.3335
Plus 100 mesh	0.2351	0.2928
Plus 150 mesh	0.2170	0.2861
Plus 200 mesh	0.2265	0.2714
Minus 200 mesh	0.2493	0.3161

All fine waste tailing is discharged into one of two 42-in. tailing flumes and carried to the beach, a distance of about 3,000 ft. Grade of the old flume, with concrete lining, is 4 per cent. The new flume, lined with 2-in. cast-iron liners cemented in place, has a grade of 2½ per cent. A few hundred feet of rubber lining was installed in the new flume for experimental purposes, but has now been replaced by iron liners. This flume has been in service for about three years and the liners are still in good condition, being worn from ⅜ to ¾ in. A dilute pulp, about 10 per cent solids, is required on account of the high percentage of coarse sand in the tailing. A typical screen analysis of tailing is given in Table VIII.

Table VIII—Screen Analysis of Tailing

Size	Per Cent	Cumulative Per Cent
Plus 10 mesh	4.3	4.3
Plus 14 mesh	9.9	14.2
Plus 20 mesh	16.8	31.0
Plus 28 mesh	13.6	44.6
Plus 35 mesh	13.8	58.4
Plus 48 mesh	9.5	67.9
Plus 65 mesh	5.7	73.6
Plus 100 mesh	6.0	79.6
Plus 150 mesh	3.4	83.0
Plus 200 mesh	6.3	89.3
Minus 200 mesh	10.7	100.0

Shipping concentrate produced on the re-treatment floor is caught in boxes at the end of the tables and packed in double burlap sacks. Concentrate produced on the Wilfley floor is conveyed in launders to the concentrate tanks, which are provided with filter bottoms and sides; after draining in the tanks, it is shoveled onto a platform, allowed to dry for two or three days, and then packed in paper-lined burlap sacks. Average moisture content is about 5 per cent. Assay content averages about $250 to $300 per ton gold, 30 oz. silver, 56 per cent lead, and 20 per cent iron. The high gold content of the concentrate is the result of inclusion of very thin flat particles of gold, which tend to ride on top of the galena. Flotation concentrate, because of its coarseness, is settled in a small tank, dewatered, and drawn out onto the floor. It is then packed in double burlap sacks and allowed to stand until a shipment is ready. Moisture content is about 10 per cent. Assay content averages about $100 per ton gold, 15 oz. silver, 20 to 25 per cent lead, 40 per cent iron. Table concentrate is shipped each ten days, and flotation concentrate about once a month.

High-grade gold concentrate, assaying $20,000 to $30,000 per ton in gold, is collected each day in an amalgam barrel for treatment. About 600 oz. of quicksilver, previously cleaned with a 10 per cent nitric acid solution, and about 3 lb. freshly slacked lime, are added to the charge, and all are ground for five or six hours. The amalgam barrel, of cast iron, 2x4 ft., is revolved 30 r.p.m. and has one piece of 6-in. shafting as a grinder. The shafting has been found more satisfactory than balls; it keeps the inside of the barrel smooth and facilitates removing all the amalgam after each run. The barrel is discharged, while running, through a ½-in. hole, into a 4-ft. mechanical batea, which removes most of the galena. Residue in the batea is carefully cleaned by hand, and the cleaned amalgam is squeezed in an air-operated press. Value of the concentrate discharge from the batea averages about $75 gold per ton.

Amalgam is retorted at regular intervals in an oil-fired, cast-iron, horizontal cylindrical retort, in small semicircular cast-iron boats. Sponge-gold recovery is 60 to 65 per cent of the weight of the

Table IX—Assays and Recoveries, 1931

	Weight		Gold	
	Tons	Per Cent	Assay, per Ton	Per Cent of Total
Rock trammed	4,162,350	100.00	$1.12198	100.00
Coarse waste	1,863,352	44.77	0.18343	7.32
Ore, fine, milled	2,298,998	55.23	1.88267	92.68
Fine tailing	2,295,768	55.15	0.26663	13.10
Bullion	.....	.....	1.24635	61.36
Lead concentrate	3,230	0.08	0.37007	18.22
Total recovery	.....	.....	1.61642	79.58
Total tailing	4,159,120	99.92	0.22936	20.42

Table X—Cost Analyses of Fine Milling

	1928	1929	1930	1931
Coarse crushing	$0.0333	$0.0313	$0.0305	$0.0262
Sorting	0.0122	0.0124	0.0136	0.0138
Screening	0.0025	0.0017	0.0014	0.0007
Intermediate crushing	0.0190	0.0178	0.0131	0.0072
Fine grinding	0.1606	0.1649	0.1791	0.1556
Concentrating	0.0221	0.0223	0.0181	0.0162
Re-treatment	0.0179	0.0147	0.0171	0.0147
General operating and maintenance	0.0588	0.0494	0.0478	0.0416
Total, per ton	$0.3264	$0.3145	$0.3207	$0.2760
Tons milled	1,790,491	2,020,470	2,066,241	2,299,018

Table XI—Cost per Ton Actually Handled

	1928	1929	1930	1931
Coarse crushing	$0.0333	$0.0313	$0.0305	$0.0262
Sorting	0.0394	0.0339	0.0368	0.0327
Coarse waste disposal	0.0432	0.0551	0.0372	0.0655
Coarse screening	0.0030	0.0022	0.0017	0.0009
Intermediate crushing	0.0739	0.0629	0.0514	0.0306
Screening, Hum-mer	0.0089	0.0130	0.0150	0.0135
Ball mills	0.1613	0.1603	0.1654	0.1442
Rolls	0.0738	0.0875	0.1104	0.0880
Tube mills	0.1773	0.1718	0.2035	0.1889
Concentrating	0.0221	0.0223	0.0181	0.0160
Re-treatment	0.1974	0.1715	0.1942	0.1722
General operating and maintenance	0.0378	0.0347	0.0334	0.0303
Total on tons trammed	$0.2285	$0.2319	$0.2271	$0.2160

Table XII—Power, Labor, and Supplies per Ton Trammed

	1928	1929	1930	1931
Power	$0.0272	$0.0269	$0.0297	$0.0248
Labor	0.1085	0.1088	0.1068	0.1002
Supplies	0.0928	0.0962	0.0906	0.0910
Total	$0.2285	$0.2319	$0.2271	$0.2160
Sorting	1928	1929	1930	1931
Crew (direct)	71.61	71.73	72.00	74.87
Cost per man per day	$8.26	$8.38	$7.43	$9.09
Tons rejected per man per day	73.37	69.36	70.90	68.37
Milling				
Crew (direct)	110.46	109.45	112.08	108.56
Cost per man per day	$17.75	$16.77	$17.07	$16.45
Tons milled per man per day	46.04	50.57	50.64	57.96

amalgam, and mercury loss is about 1 per cent. Sponge gold is melted in an oil-fired Faber du Faur tilting furnace with a No. 10 graphite retort for a crucible. Soda and borax glass are used for flux, and the molds are heated and oiled with boiled linseed oil. Bullion is sampled, while molten and before pouring, by means of a graphite sample mold made from an old retort. The sponge is divided, so as to make bars of 1,200 to 1,600 oz., which are melted one at a time, about 40 min. being required for each bar after the crucible is hot. Average grade of the bullion is: gold, 850 fine; silver, 130 to 135 fine. About 80 per cent of total gold recovery is in the form of bullion. One man takes care of amalgamating, retorting, and melting.

Water for milling purpose, obtained from Gold Creek, is delivered, without pumping, through a 36-in. flume and a 20-in. wooden-stave pipe line to a surge tank above the ball-mill floor; 8,000 to 8,500 gal. per minute is required. No recovery of water is made, all of which is required to convey the fine tailing to the beach. During very cold weather, when the supply of fresh water is inadequate, salt water—more satisfactory than fresh water during extreme cold weather, because it does not freeze so readily—is pumped from Gastineau Channel. Part salt water does not effect the flotation operation; but with all salt water difficulty is experienced in getting a clean concentrate.

Table XIV—Power Required in Each Milling Operation per Ton Handled

	Kilowatt-Hours
Coarse crushing	0.4806
Sorting	0.2586
Coarse waste disposal	0.2268
Screening (Niagara)	0.0130
Intermediate crushing (Symons cone)	0.6036
Screening (Hum-mer)	0.1991
Ball mills	9.5295
Rolls	2.9771
Tube mills	8.4848
Ball mill (tube-mill floor)	5.7266
Concentrating	0.2088
Re-treatment	0.2088
General mill operating	0.9104
Assaying and sampling	0.0230
Total on tons trammed, kw.-hr.	6.2200

Supplies for the mill and concentrates for shipment are handled by an incline tram, which extends from the dock to the crusher floor. Ball- and tube-mill floors are each equipped with a 50-ton Shaw electric traveling crane; and the crushing plant has a 20-ton crane of the same make. Monorail cranes and crawls are used on other floors.

From 160 to 165 men are employed in the mill, as follows: two crushing and sorting crews of 35 each, three operating crews of 12 each, one general repair and maintenance crew of about 40, and a general mill crew of 16 men. Assays and recoveries are presented in Table IX. Costs for the various stages of fine milling for the last few years, based on the total tonnage of fine ore milled, are given in Table X. Costs per ton handled in each operation, based on the tonnage actually handled in the operation of fine milling, are given in Table XI. Milling costs for power, labor, and supplies, based on tonnage trammed, are given in Table XII. Cost and tonnage handled per man per day for sorting and milling ore are given in Table XIII. Power consumption per ton handled in each operation for 1931 is given in Table XIV.

Mechanical Operations and Rock Disposal

The company's properties, exclusive of one power plant, are along both sides of Gastineau Channel, and include the former property of the Alaska Treadwell Gold Mining Company, on Douglas Island, which was acquired in 1928. Power is obtained from hydro-electric sources, with a steam plant serving as an auxiliary unit. The hydro-electric plants are dependent on stream flow, as water-storage facilities are limited. Heavy precipitation of the district assures a steady flow during eight or nine months of the year, and in the other months any additional power required is generated at the steam plant or purchased from outside sources. . . . Underground ventilation is important, as a large amount of explosives is used daily throughout the mine, particularly in secondary breaking. In bulldozing areas, ventilation raises connect with the surface and motor-driven fans installed. . . . Underground pumping was not necessary until a year ago, when water encountered in sinking a winze could not be directed by gravity methods to a drain tunnel, through which the rest of the mine water flows to waste on the surface. A salt-water plant pumps water from Gastineau Channel to the concentrator building for milling purposes. . . . The machine shop and boiler shop are in the concentrator building, conveniently situated for handling most of the heavy equipment on the property. Two electric furnaces are used in the foundry, on Douglas Island, where castings of steel, cast iron, and brass supply renewal and repair parts.

J. A. Williams
Chief Engineer

Alaska Juneau holdings, amounting to nearly 1,500 acres, may be grouped as follows: the beach division; mine division; Douglas Island holdings; outlying hydro-electric plants. The beach division is on the east side of Gastineau Channel and joins the southern boundary of the city of Juneau. It includes wharves and warehouses, main office building, auxiliary steam power plant, 11,000-ton concentrator, carpenter shop, and coarse waste-rock disposal plant. The larger buildings, such as the steam plant and the concentrator, are of structural-steel frames, with corrugated-iron sides and roof. Concrete construction is, of course, used extensively in these buildings. The warehouses and most of the other buildings have corrugated-metal roofs and sides, supported by timber framing. This is a satisfactory type of building for the locality. The relative positions of these structures is shown in the two maps that appear on pages 464, 486, and 487.

The mine camp is in Gold Creek valley, about 1½ miles northeast of the beach plant. This community, known as Jualpa, is commonly called the "Basin." Locomotive- and car-repair shops, the compressor plant, and a dormitory with boarding house to accommodate one hundred men are at this point. The camp was established in 1911 in connection with the driving of Gold Creek tunnel, the portal of which is part of the Basin camp. Beginning in Gold Creek adit, at a point 4,000 ft. from the portal, the mine extends several thousand feet both southeasterly and northwesterly. The concentrator, served by the main tram operating

through Gold Creek tunnel, through snowsheds on the side hill, and through three short tunnels, is about 2½ miles from the mine. Elevation of the main level of the mine is about 440 ft. above sea level.

The Douglas Island property is on the west side of Gastineau Channel, 2 miles southwest of Juneau, and consists of the holdings of the Treadwell group of mines, which were acquired in 1928.

Average Precipitation, Snowfall, and Mean Temperature

	Jan.	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Annual
Total precipitation, in. (*a*).....	7.04	5.49	5.59	5.38	5.26	3.58	4.89	7.30	10.40	10.98	8.54	7.64	82.09
Snowfall, in.....	31.7	22.5	12.9	5.3	.....	.....	.....	.....	.....	1.4	7.3	30.3	111.50
Mean temperature, deg. F.....	26.9	29.7	33.4	40.8	47.9	54.2	57.3	55.0	50.1	43.0	34.9	30.9	42.0

(*a*) Includes melted snow. At Juneau, 13 in. of snow is equivalent to 1 in. of water.

One hydro-electric power plant and the foundry are situated on the property. Two outlying hydro-electric power plants, the Sheep Creek plant and the Nugget Creek plant, are situated 4 miles south and 14 miles northwest of Juneau, respectively.

Power for the use of the various departments is obtained from three sources: the Alaska Juneau hydro-electric system, the auxiliary steam plant, and the Alaska Gold hydro-electric system. Alaska Gold is the holding company for the Alaska Gastineau Mining Company, which has not operated since 1921. For this reason most of the power from the two Alaska Gold hydro-electric plants is for sale. The power plants and transmission lines are shown in the accompanying map.

Alaska Juneau's hydro-electric plants were acquired from the Alaska Treadwell Gold Mining Company in 1928, from which company power was previously purchased. The plants are almost entirely dependent on natural flow of streams, as water storage is either impossible or too expensive. Copious rains, melting snow, and ice, however, afford a steady flow of water for eight or nine months of the year. During the remaining months the deficiency is made up by purchase of power from the Alaska Gold company or by generating power at the company's auxiliary steam plant. Meteorological data for Juneau are presented in the accompanying table.

Among the assets of the Treadwell company taken over by Alaska Juneau in 1928 is the "240" hydro-electric plant, rated at 1,000 kw. and operating under a static head of 460 ft. Water for this plant is taken from a 14-mile system of ditches that was started in 1882 and gradually extended and improved until all the mills on Douglas Island derived power from the system during the run-off season.

The generator installed in the "240" mill is directly connected to a 1,400-hp. Pelton-Doble double overhung impulse water wheel, with one nozzle on each wheel and a Pelton Size O governor. The unit was intended to be used either as a generator or as a motor, the connection to the mill being through a rope drive from the generator shaft. Inasmuch as the Treadwell group of mines is no longer in operation, the "240" mill has been dismantled and the power plant housed in a timber-frame corrugated-iron structure. The Pelton-Doble water wheels are mounted on the same shaft with the 1,012-kva., 2,300-volt, 400-r.p.m., 3-phase, 60-cycle Westinghouse generator. The exciting set consists of a Pelton waterwheel directly connected to a 38-hp., 850-r.p.m., 2,200-volt Westinghouse motor that drives a 25-kw., 125-volt, 850-r.p.m. Westinghouse generator. Power is stepped up for transmission to 23,000 volts by means of three 500-kva. Allis-Chalmers water-cooled transformers, delta-delta connected.

The Sheep Creek plant, with a rated capacity of 2,300 kw., is on tidewater at the mouth of Sheep Creek, 4 miles south of Juneau. A 4x4-ft. timber flume 2,500 ft. long leads the water of Sheep Creek to a 2,675-ft. penstock of 36-in. and 34-in. steel pipe that delivers the water to the power plant. Static operating head is 600 ft. The power house, which is of timber frame and corrugated iron covering, houses three hydro-electric units. Two identical sets consist of a 1,900-hp. Pelton impulse wheel, with two nozzles operating at 400 r.p.m. and controlled by a Pelton Type 0-1 governor, directly connected to a 1,000-kw. at 85 per cent power factor, 3-phase, 60-cycle, 2,300-volt General Electric generator. Each of the two sets of exciters include one 25-kw., 125-volt, 1,200-r.p.m. G.E. generator, directly connected to a 35-hp., 2,300-volt, 1,200-r.p.m. G.E. motor, which, in turn, is directly connected to a Pelton waterwheel. The third unit includes a 225-kw. at 90 per cent power factor, Westinghouse generator, operating at 3 phase, 60 cycle, 2,300 volts, and 400 r.p.m., set between two waterwheels. One wheel is a 300-hp. Pelton-Doble unit that operates under a 600-ft. head. The other is a similar wheel and operates under a 270-ft. head. Both are controlled by a Pelton Type 0-1 governor. Water making up this 270-ft. static head is led from Sheep Creek through what is known as the low-pressure flume and penstock. The transformer station includes three G.E. self-cooled transformers, with a capacity of 2,000 kva., 2,300 to 22,000 volts.

Nugget Creek power plant is adjacent to Mendenhall Glacier, about 14 miles northwest of Juneau. The drainage basin of Nugget Creek has an area of 14.6 square miles and includes a small hanging glacier. Near the lower end of the valley, a crib dam forms a small reservoir. To sluice out the sand that quickly accumulates behind the dam, an 8x8-ft. concrete tower with removable plank panels has been constructed in the reservoir. The tower has a tunnel outlet to the creek below the dam. Water from the forebay is diverted through a 650 ft. tnnnel, 7x8½ ft. in cross-section, and thence through a 48-in. wood-stave pipe 4,562 ft. long, built partly on trestle and partly on the ground to a point 490 ft. above the power house. From this point, a 36-in. steel penstock, 2,020 ft. long, delivers the water to the power house.

The power house is a frame building, with concrete foundations and floor, covered with corrugated steel. The first unit installed includes a 1,900-hp. single-runner overhung Pelton-Doble impulse wheel, with two nozzles 5¼ in. diameter, operating at 300 r.p.m., controlled by a Type Q Lombard governor, and directly connected to a 1,000-kw. at 85 per cent power factor, 3-phase, 60-cycle, 2,300-volt G.E. generator. For this unit, the excitor is a motor-generator waterwheel set, including one 18 kw., 125-volt, 1,200-r.p.m. G.E. generator, directly connected to a 35-hp., 2,300-volt, 1,200 r.p.m. G.E. motor, which, in turn, is directly connected to a Pelton-Doble waterwheel, without a governor. The second unit includes two single-runner, 1,900-hp., Pelton-Doble impulse wheels, with two needle nozzles per wheel, operating at 400 r.p.m. on the same shaft, controlled by one Lombard governor, and directly connected to a 2,350-kva., 3-phase, 60-cycle, 2,300-volt, Westinghouse generator. For this unit, the exciter is a 50-kw., 125-volt, 680-r.p.m. Westinghouse generator, directly connected to a 75-hp., 2,300-volt, 680-r.p.m. Westinghouse motor, which, in turn, is directly connected to a Pelton-Doble waterwheel, without a governor. A third exciter set at this plant includes a 40-kw., 120-volt, 650-r.p.m. Bullock generator, directly connected to a Pelton-Doble waterwheel, without a governor.

According to the original layout, completed in 1917, the steam plant was the main source of power for Alaska Juneau, because at that time the Treadwell group of mines and the Alaska Gastineau company required all of the

hydro-electric power available. However, after the Treadwell cave-in, which ended the life of three of the Treadwell mines, to use the available hydro-electric power was more profitable than to employ steam-generated power, so the steam plant became a standby. With the elapse of time power requirements of the Alaska Juneau have increased greatly. Besides, hardly a winter passes without a power interruption of several weeks' duration, as a result of a snowslide or sleet on the transmission line of the Alaska Gold company where it crosses a high range of mountains. For these reasons the steam plant is indispensable during the three or four winter months.

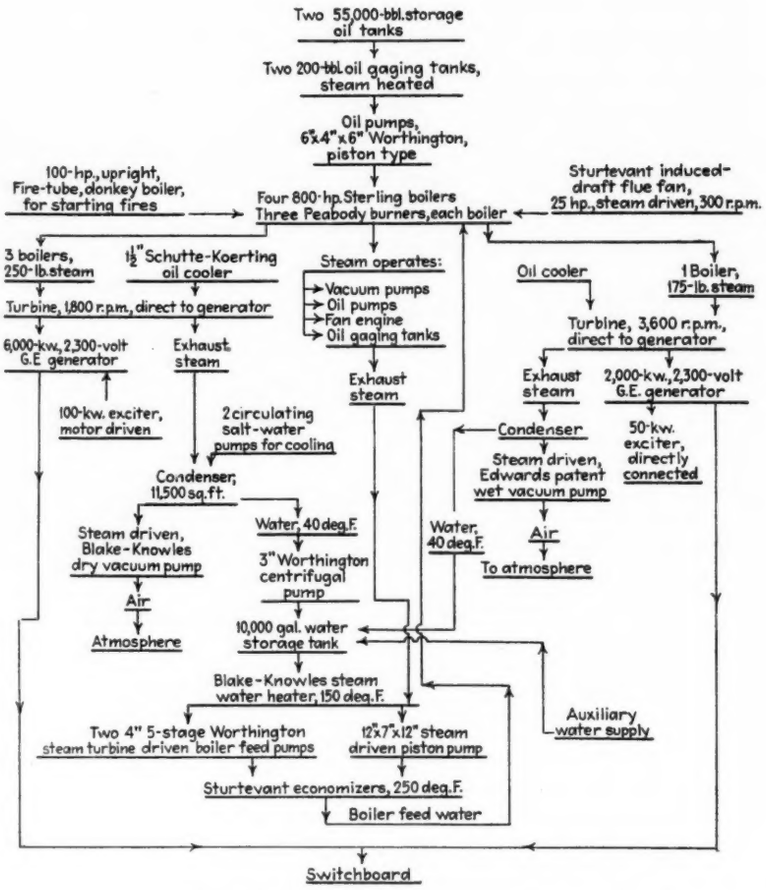
The steam plant, as shown in the accompanying flowsheet, consists of four oil-fired 800-hp. Stirling boilers that supply superheated steam to two turbo generators, of 6,000-kw. and 2,000-kw. capacity, respectively. Equipment includes condensers, economizers, fuel-oil heaters, vacuum pumps, and draft fans. The steam power plant is the main distributing station, and all power lines from outlying power plants therefore enter the building. An accompanying illustration shows graphically the general arrangement of the electric power distribution system.

From the northwest, a line 13 miles long, of three-conductor, No. 00 copper, enters the plant. This line is supported by Pyrex glass, pin-type insulators, attached to native yellow-cedar poles, 40 ft. high. Although no preservative treatment is given the poles, in some sections they are covered with sheet iron to a height of 3 ft. above the ground for protection against porcupines. Average life of the poles is about ten years. Western red cedar poles are used for replacement, as they have a longer life.

Previous to 1926 all 23,000-volt power lines were carried on old-type insulators, with little or no provision for expansion and contraction. An excessive number of power failures, as many as three in one day, occurred on hot summer days, owing to cracked insulators. During the winter of 1926, all strain-type insulators were replaced, and since then all pin-type insulators have been provided with Pyrex glass mounted on lead thimbles. This has completely eliminated power failures resulting from line trouble during the last three years.

By means of a test set provided with ear phones, an inspection, with the lines charged, is made once a month during the summer, and in the winter, at times when the line is dead, a more thorough examination is made.

One mile from Juneau the Nugget Creek line joins the Douglas Island transmission line, which is 3 miles long and of the same general type. A span of 940 ft. is necessary to cross the narrow part of Gastineau Channel. At the lowest point of the span the line is 50 ft. above high tide. Incoming power from the Sheep Creek plant follows the east side of Gastineau Channel. Jualpa Basin and the mine itself are supplied with power from a 2-mile branch of the main transmission line. Both of these lines are supported in the same manner as the aforesaid lines.


Two 55,000-bbl. storage oil tanks
Two 200-bbl. oil gaging tanks, steam heated
Oil pumps, 6"x4"x6" Worthington, piston type
100-hp., upright, Fire-tube, donkey boiler, for starting fires
Four 800-hp. Sterling boilers Three Peabody burners, each boiler
Sturtevant induced-draft flue fan, 25 hp., steam driven, 300 r.p.m.
3 boilers, 250-lb. steam
½" Schutte-Koerting oil cooler
Steam operates:
Vacuum pumps
Oil pumps
Fan engine
Oil gaging tanks
Oil cooler
1 Boiler, 175-lb. steam
Turbine, 1,800 r.p.m., direct to generator
6,000-kw., 2,300-volt G.E. generator
Exhaust steam
Exhaust steam
Turbine, 3,600 r.p.m., direct to generator
Exhaust steam
2,000-kw., 2,300-volt G.E. generator
100-kw. exciter, motor driven
2 circulating salt-water pumps for cooling
Condenser
50-kw. exciter, directly connected
Condenser, 11,500 sq.ft.
Steam driven, Edwards patent wet vacuum pump
Steam driven, Blake-Knowles dry vacuum pump
Water, 40 deg.F.
Water, 40 deg.F.
Air
3" Worthington centrifugal pump
To atmosphere
Air
Atmosphere
10,000 gal. water storage tank
Blake-Knowles steam water heater, 150 deg.F.
Auxiliary water supply
Two 4" 5-stage Worthington steam turbine driven boiler feed pumps
12"x7"x12" steam driven piston pump
Sturtevant economizers, 250 deg.F.
Boiler feed water
Switchboard

Flowsheet of auxiliary steam power plant

Three substations are employed: the central, the Basin, and the mine substations. The central substation is at the steam plant on the beach. A bank of three Westinghouse self-cooled transformers, having a total capacity of 4,000 kva., steps the voltage down from 23,000 to 2,300 volts. A separate bank of three water-cooled transformers serve as spares. The substation is fully equipped for distributing power to the various plant operations. The Basin substation is at the compressor building in Jualpa Basin. A bank of General Electric self-cooled transformers, having a total capacity of 2,000 kva. from 23,000 to 2,300 volts, and a bank of three Westinghouse water-cooled transformers, having a total capacity of 900 kva. from 23,000 to 2,300 volts, are employed with the various circuits radiating from the station. A 300-kva. rotary converter unit, complete with 23,000- to 440-volt, 6-phase, 60-cycle transformers, delivers 550 volts d.c. into the trolley system. The mine substation is equipped with switching panels, tie-ins, and circuit breakers. A 300-kva. rotary converter unit, with a 2,300- to 440-volt transformer, is also included in the equipment.

Power consumed in 1931 totaled 36,235,663 kw.-hr. of which 70 per cent was generated by the company's hydro-electric plants, 29 per cent was purchased from the Alaska Gold company, and 1 per cent was generated in the auxiliary steam plant. Average requirement per hour was 4,150 kw.; average peak load handled by the whole system, 6,000 kw.; average cost per kilowatt-hour, $0.00378; cost of power consumed per ton trammed, $0.0329; kilowatt-hours per ton (total), 8.71; kilowatt-hours per ton trammed underground, 1.39; kilowatt-hour per ton for surface tramming, 0.66; kilowatt-hours per ton for milling, 6.22; kilowatt-hour per ton for general expense, 0.44. Distribution of power consumption by departments in 1931 is shown in the accompanying chart.

Power to operate the trolley system can be taken from either the 2,300-volt supply line at the Central station on the beach, the substations at the Basin camp, or the underground substation at the mine end. Two of these sources are used, and the third source is held in reserve. Power is delivered to the trolley wire at 550 volts, d.c., through 300-kw., 6-phase, 60-cycle, rotary converters. To serve the substation in the mine, two 2½-in. three-conductor submarine cables bring in 2,300 volts, a.c., from the main power line. Between the Central substation and the mine end, a 4/0 trolley feeder helps to maintain the voltage if the mine rotary converter is not operating. No. 3/0 round and No. 3/0 grooved copper trolley wires are used, supported every 25 ft. by standard clamp-and-span wire-type hangers. The trolley wire is vertically above one rail, at a height of at least 7 ft. 3 in. Trolley boards are not used except at chutes and crossings. Trolley frogs are in use in many places but not throughout the system.

In the mine, for handling sections of the main train, three 18-ton General Electric locomotives made up of two 9-ton units are available, and also one 10-ton Jeffrey locomotive. Both types have solid frames. The motors are d.c. series-wound inclosed-type units, with ball bearings. The 18-ton articulated locomotives are equipped with four 40-hp. motors that drive 30-in., steel-tired wheels, whereas the 10-ton locomotive has two 50-hp. motors driving 33-in., steel-tired wheels. Two 6-ton G.E. trolley locomotives, each driven by two 25-hp. motors, are maintained for hauling supplies, handling the shift train, and for general utility work. Three 18-ton Baldwin Westinghouse locomotives, with bar-steel frames, are in commission for hauling ore from the mine to the tipple. They are made up of two 9-ton units in tandem, using four 31-hp. motors geared to 33-in., steel-tired wheels. Solid spur gearing, with a thirteen-tooth pinion and an 86-tooth gear, is used in preference to helical gears. They are fully equipped with Type M-22-A standard automatic air brakes. This air-brake system also includes the first four cars on all outbound trains. Each Westinghouse haulage locomotive uses continuous forced ventilation for the motors, which is produced by a No. 1½ Sirocco blower.

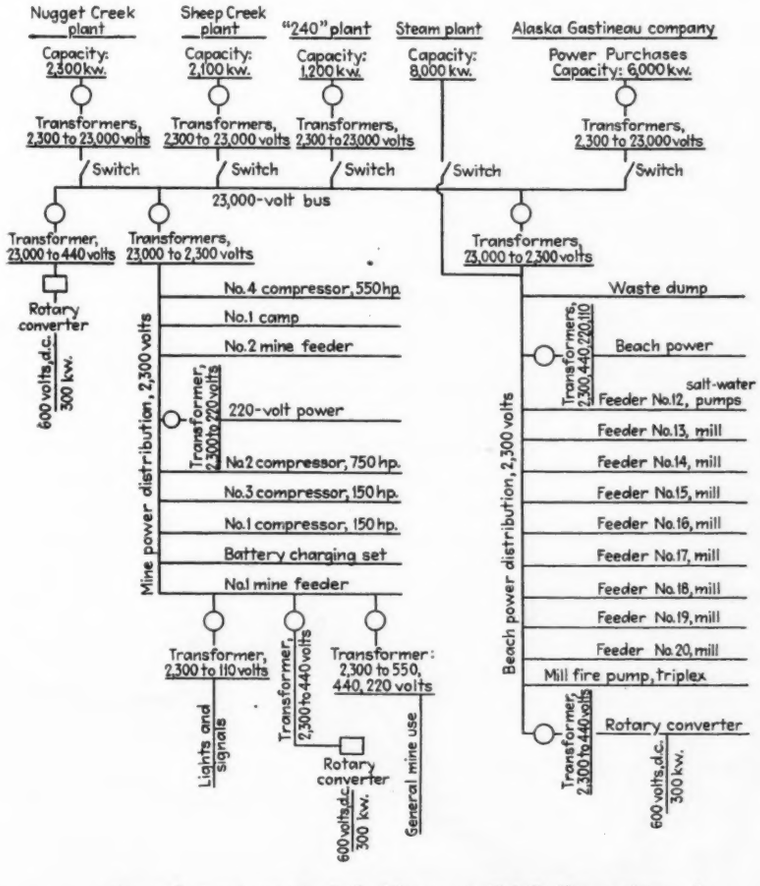
All trolley locomotives are equipped with side-mounted trolley poles. No lightning arresters are used, and leads are continuous from trolley to controller. The original grid resistance has been discarded for Type M edge-wound resistance, which has a high tensile strength.

On the main level of the mine, two 5-ton and one 6-ton Edison storage-battery locomotives are used. A charging period of fou. hours is generally sufficient for the work they do in eight hours. One charging set is maintained in the mine.

The compressed-air plant for the mine consists of four units, all built by the Ingersoll-Rand Company. Until recently the principal unit was a Class PRB, 22½x36x27-in., 150-r.p.m. compressor, capable of compressing 4,000 cu.ft. of free air per minute to a final pressure of 100 lb. at sea level. This machine is equipped with a 1,512-HM-2 aftercooler. A 2,300-volt, 3-phase, 60-cycle, 750-hp. induction motor, operating at 355 r.p.m., drives the compressor through a 1⅜-in. manila-rope drive, consisting of twenty turns around pulleys at 35-ft. centers. The company recently installed a PRE-2 compressor, with high-pressure cylinder 18 in. in diameter, low-pressure 30 in., and stroke 21 in., and having a capacity of 2,550 cu.ft. of free air per minute. A 500-hp., 2,300-volt, 3-phase, 60-cycle Westinghouse synchronous motor, directly connected and turning at a speed of 180 r.p.m., furnishes the power. Through an aftercooler, Type 1210-HM-2, the air is discharged into a 300-cu.ft. receiver, from which it is led into the mine through a 10-in. air main. This main, welded into 100-ft. lengths, is connected by Dayton couplings. Of the two remaining compressors, one is an Imperial, Type No. 10, with a capacity of 750 cu.ft. per minute; the other is of the same type, with a capacity of 600 cu.ft. free air per minute. Both are belt driven, the first by a 200-hp. wound-rotor type induction motor, and the second by a 150-hp. motor of the same type.

So far, hoisting in the mine has required but little power. Of the two principal manway raises that extend above the main level, one is served by a single-drum geared hoist driven by 40-hp., d.c. motor, operating off the trolley circuit. The second raise is served by an air hoist. In both installations the hoist is set at the bottom of the raise.

Electric hoists are used in the two prospect winzes. They are gear-driven by induction motors of wound-rotor type. In No. 53 winze, a 100-hp., single-drum hoist operates a 2-ton skip to No. 10 level, a distance of 1,150 ft., at an inclination of 60 deg. Sinking below this level is continued by means of a 6x8-in. air hoist. In No. 91 winze, where sinking alone is done, a double-drum, 50-hp. hoist is used. As part of the program for lateral development work and preparatory mining in



General arrangement of electric power distribution system

this vicinity, a double-drum, 150-hp. hoist has recently been installed at the top of No. 91 winze.

Inasmuch as 80 cases of powder per day are used, and secondary breaking accounts for three-quarters of the total explosives consumed, good ventilation is important. Owing to the general arrangement of major development openings and their size, and to the generally cool outside temperature, the natural ventilation through the mine is excellent. To stimulate the draft in bulldozing areas, ventilation raises have been driven to connect with the surface, and 60-in. Ventura-type fans, belt-driven by 15-hp. motors, have been installed. Four of these installations in the south half of the mine, and one in the north half, clear out the powder smoke to a remarkable degree. For winze sinking, 10-hp., direct-connected, blower-type exhausters are in use.

Until a year ago, pumping was not necessary at the Alaska Juneau, as all water drained through Gold Creek tunnel. At present a 3½x8-in. vertical Triplex pump, geared to a 20-hp. motor, handles the water in No. 53 winze. In the sinking of No. 91 winze, pumping has not as yet been necessary.

Power is distributed throughout the Beach division from the central substation switchboard. The general arrangement of power distribution is shown in an accompanying diagram. The eight separate feeders for mill operations are housed in a cableway, whereas power lines for the mill fire pumps follow a more circuitous route that affords less danger from fire. In the mill the tipples and traveling cranes are operated by d.c. motors. All other equipment, however, is operated by a.c. motors.

Each of the two full-revolving, four-car tipples are operated by a 70-hp., 550-volt, d.c. motor. To reverse the tipple the current is reversed. Although not ordinary practice, this has proved satisfactory. Generally, all motors in the surface division, up to and including those of 15 hp., are operated at 220 volts, whereas 2,300 volts is used on all motors above 15 hp. The last 2,300-volt motors installed have controls that consist of air-break contactors for across-the-line starting. The older starters are the Westinghouse Type I oil switches.

In the coarse-crushing plant 150-hp., squirrel-cage induction motors drive the 36x48-in. jaw crushers. The 9-K gyratories are driven by 100-hp. motors of the same type. Some of the coarse-crushing units are driven from a line shaft connected to a 350-hp. synchronous motor. Motors driving the 4-ft. cone crushers are 100-hp., squirrel-cage, induction-type units, equipped with ball-bearings, as they operate at a speed of 1,800 r.p.m.

Fine crushing, as indicated in an accompanying chart for 1931, is a great consumer of power. This operation consists largely of ball milling, which involves the reduction of minus-2½-in. material to minus 7 mesh. Eleven ball mills, each belt driven by a 225-hp., squirrel-cage induction motor, are operated. Recently, two units directly connected to 225-hp. synchronous motors have been added. The tube mills and rolls in the fine-grinding department are belt driven by 150-hp., squirrel-cage induction motors.

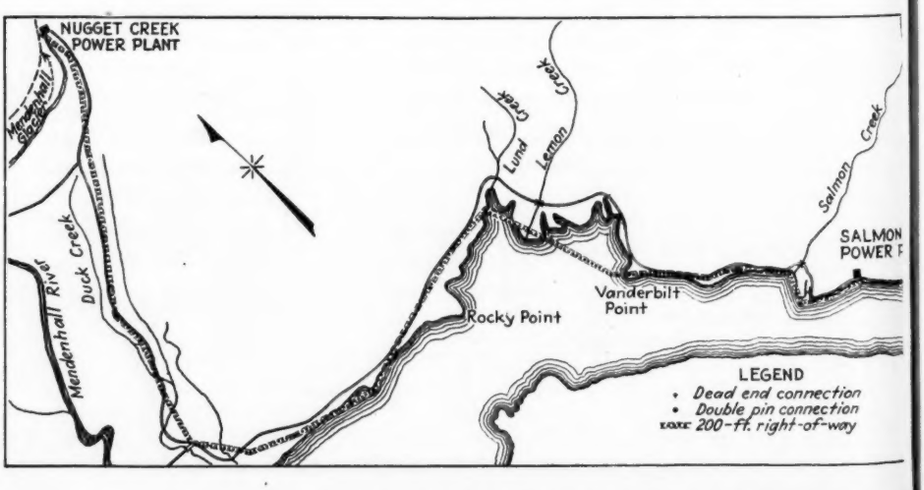
A salt-water pumping plant, on the lowest level of the steam power plant building, is used for pumping water from Gastineau Channel to the concentrator building, where it is used in milling and metallurgical operations to supplement fresh water that is carried in flumes from Gold Creek. In the winter, this salt water often becomes the sole supply for the concentrator. The mill water and pumped salt water are also used for a second fire line system for the city of Juneau. When the steam plant is in operation, two 270-kva., synchronous, motor-driven circulating pumps supply salt cooling water for the condensers. For the mill water supply three 3-stage centrifugal pumps, with 10-in. intake, each having a capacity of 3,000 g.p.m. against a 273-ft. head, are driven by a 400-hp., squirrel-cage induction motor.

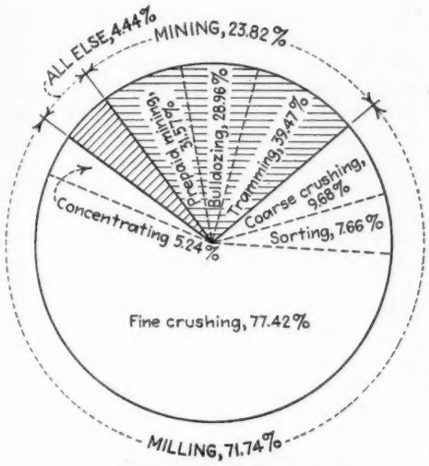
In the disposal of coarse waste a small amount of power is used in loading and starting the cars down the incline. Once started, the loaded train pulls the empty train up, and also causes power to be fed back into the line. A slight net gain in power is shown in the operation. The double-drum hoist is geared to a 150-hp., wound-rotor type, induction motor. The main conveyor and the five branch stacker conveyors are each equipped with tandem drives, to which are connected 75-hp., squirrel-cage induction motors. For the second waste dump unit, 550-volt d.c. power is used on the trolley, the 9-ton locomotives being duplicates of the 9-ton units used in the mine as 18-ton articulated locomotives.

Car repair shops and locomotive repair shops are on sidings near the main haulage line at Jualpa Basin. Double tracks, equipped with pits, run throughout the length of each building. The car shop is 30x110 ft., with a wing 30x40 ft., to accommodate the blacksmith shop. The building is of wood frame with corrugated-iron covering. In the car shop the following work is done: repair and maintenance of 190 10-ton ore cars; repair and fabrication of 4½-ton Granby cars; repair and fabrication of skips, buckets, scrapers, chute gates, and miscellaneous mine equipment; fabrication of 15-in. mine grizzly beams; miscellaneous electric and oxy-acetylene welding. The shop is equipped with cranes, monorails, car-straightening devices, air hoists, air hammers, power saw, welding and cutting outfits, and other apparatus necessary for the efficient handling of all operations.

Repair work in the locomotive shop, largely of an electrical nature, has as its object the repair and maintenance of all electric locomotives used throughout the plants.

Previous to 1925, maintenance on trolley locomotives was more nearly repair work than it is today. For instance, when an armature burned out a new armature was put in to replace it, and no special check or test of the rest of the equipment was made to ascertain the cause of the trouble. At one time, during a winter colder than usual, eight armatures were burned out on haulage locomotives and six on loading locomotives in one month. Since then a regular inspection is made once every six months, at which time each locomotive unit is dismantled. The armatures and field coils are removed from the motors, and then tested, dipped, and baked; all bearings and brake equipment are inspected; all wiring and mechanical parts thoroughly overhauled and repaired. In the last three years





TOTAL POWER CONSUMPTION IN 1931
36,235,663 KW.-HR.

Power consumption chart for 1931.

that this system of inspection has been used, one armature has burned out on the hauling locomotives and five armatures have burned out on the loading locomotives. All classes of locomotives are subjected to a daily inspection, besides a tearing down and thorough overhauling every six months. Other duties of this department are: maintenance of the whole trolley system, including the lights and signals along the haulageway; maintenance and repair of all electrical equipment in Jualpa Basin; shop repairs to mine electric hoist and fan motors, but not their maintenance; cooperation with mine electricians.

The machine shop has since 1923 occupied what was known as the roughing-table floor in the original 8,000-ton concentrator building. Before that time the machine shop and boiler shop were at Treadwell, which necessitated ferry service and caused some delay. The present location of both of these shops in the concentrator building is convenient, because a minimum of handling is involved in connection with the heavier work, nearly all of which is for some department of the mill. The machine shop, 50x210 ft., is separated from the rest of the mill by a dustproof and nearly soundproof partition. A pipe-threading and pipe-cutting department is housed separately in a 20x30-ft. addition.

Of the total machine-shop work, which, as a rule, is of a commonplace nature, mill repairs comprise 70 per cent. Occasionally, however, a 9-K crusher shaft will require rebuilding, rethreading, and turning down; or 36x48-in. jaw crusher parts will need repairing. By making the stationary jaws in sections, both cost of repairs and cost of renewals have been reduced. The remainder of the work consists largely of: turning out 10-ton car axles, and fitting them with new wheels; making cylinders for air-operated chute gates; turning down conveyor pulleys; and miscellaneous shop work.

Equipment consists of the usual lathes, drill presses, shapers, key seaters, hydraulic press, and other apparatus. For handling heavy lathe work up to 7 tons, excellent results are obtained with a 44x20-ft., heavy-duty engine lathe, driven by a 30-hp., 870-r.p.m. motor, with a separate 2½-hp. motor for traverse motion. The shop is well served by two electric cranes, of 5 and 10 tons' capacity, respectively, besides a 4-ton monorail crane. A force of 21 men is employed in the machine shop.

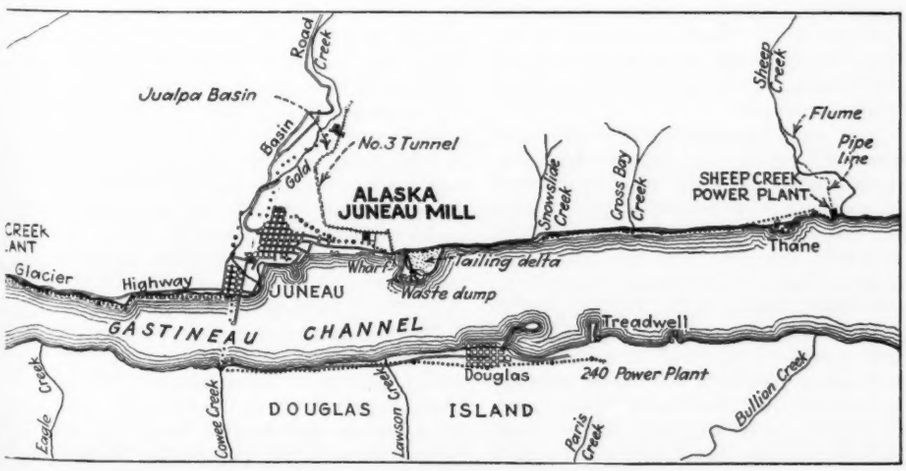
The boiler shop, 60x70 ft. in area, is also housed in the mill building, on the floor below the machine shop. Besides having its own 5-ton electric crane, it is served by the 40-ton tube-mill electric crane. Both the machine shop and boiler shop are alongside the mill service tram, which gives ready access to the wharves and other departments. A crew of eight men is employed in the boiler shop on the following: sheet-metal work; cars and trucks; mine grizzly beams; tube-mill and ball-mill scoops; ventilation pipe for mine; tube-mill and ball-mill repairs; cutting and welding of all classes. Gyratory mantles are regularly built up and hard-faced with Hascrome by electric welding. Stationary and movable jaws of 36x48-in. crushers are also built up with this chromium-manganese-iron alloy. Both the electric and oxy-acetylene welding sets are used a great deal for welding and building up various other pieces of equipment.

The foundry, in Treadwell, on Douglas Island, was erected and operated by the Alaska Treadwell Gold Mining Company for many years. Its cupolas were later superseded by the present 2-ton Heroult electric furnace, and other changes were made as time went on. The foundry was among the Treadwell holdings taken over by Alaska Juneau in 1928. Equipment, besides the electric furnace, includes an annealing furnace, a brass cupola, two core ovens, a 5-ton electric crane, 350-cu.ft. compressor, shears, drop hammer, and the customary molding and cleaning apparatus. A steam-heating plant and modern dry room are also maintained. Castings of steel, cast iron, and brass are turned out. The bulk of the tonnage is included in the following: liners for tailing flumes and mill launders, pumps, junction boxes, and other parts; conveyor pulleys and idlers; 10-ton car drawheads and pedestals; liners for 5-ft. ball-pebble mill, and end liners for tube mills; steel car wheels for miscellaneous small cars.

Large quantities of scrap iron of all kinds resulting from years of the Treadwell operations, besides the current mill steel scrap, are available for remelting and re-use.

The sharpening shop for mine drill steel is on the surface at the top of No. 1 raise, which is the main manway in the south half of the mine. Formerly, the shop was underground. To expel the gases from the oil-fired furnaces, a fan and 2,000 ft. of fan pipe were necessary, and, although fairly efficient, upkeep was high and the whole arrangement unsatisfactory. The present layout necessitates hoisting fuel up the raise, but this is conveniently accomplished by means of a special tank car that discharges its load into a 10-bbl. storage tank on the surface. Contrary to general belief, a comparatively small amount of drill steel sharpening goes on at the Alaska Juneau. At present it is increasing, owing to the larger than usual development program that is in effect. The shop, 40x25 ft., is of lumber and corrugated-steel construction and is well ventilated and lighted.

Steel is handled between the top of the raise and the shop by hand trucks that operate on 18-in. gage track running the full length of the shop. The furnace, double-opening and of local design, is fired with fuel oil fed by air pressure through a Denver Fire Clay low-pressure burner. The shop is oper-



ated on a three-shift basis. One sharpener and a helper work on each shift, and on day shift the chief blacksmith and a helper prepare and temper shanks. Two drill-sharpening machines are available for making bits, and a third machine is used for shanking steel only. An emery wheel and a shank grinder are about the only accessories used.

The steel is cut into lengths by a power hacksaw in the car-repair shops. About 1,000 pieces of steel are sharpened in 24 hours. Half of this is 1⅛-in., hollow, hexagon steel, used by water Leyners and wet stopers; the other half is made up of ⅞-in., hollow, hexagon steel, used for blockholing and for shaft sinking. The rock drill bits are standard cross bits, with single taper and ¼-in. change in gage for 1⅛-in. Leyner steel, and with double taper and 5/16-in. change in gage for ⅞-in. shaft steel. The size of starters for 1⅛-in. Leyner steel is 2½ in., and 2 in. for ⅞-in. Jackhamer steel. Changes in length are 20 in. for Leyner steel and 18 in. for shaft steel. Change in length for blockholing steel varies considerably. A small amount of 1-in., solid, quarter-octagon steel is used in drilling hang-ups in the bulldozing operations. This is handled in the shop in the same manner as Leyner steel, except that dry dollies are used. The ⅞-in. hexagon steel requires a collared shank; all other steel is made with a plain shank for use in the anvil block chucks. Carbon steel is used for all drill bits. Sharpening and hardening are done in two separate heats. For hardening, the bit is heated to a bright cherry-red heat and then plunged into cool running water. Shanks are draw-tempered and plunged at the change from copper to blue color.

DISPOSAL of coarse waste at Alaska Juneau consists of taking the waste rock from the stockpile alongside the mill building, transporting it by gravity tram and a system of belt conveyors to a point 3,500 ft. south of the mill, and there discharging it into Gastineau Channel. The gravity tram has an 8 per cent grade, on which two trains of cars are operated in balance, control being effected by a double-drum hoist at the head of the incline.

Each train consists of nine 4½-ton Granby-type cars equipped with 18-in. roller-bearing wheels. Cable used is 1-in. haulage, 6 by 7 lang-lay rope. The loading station consists of six reinforced concrete drawholes, each equipped with a standard chute, 5 ft. wide, with underslung arc gate counterbalanced so as to be easily operated by one man. The train is loaded by two men, in two spottings of the cars.

The tram, mostly trestle, is double tracked with 50-lb. rail, with single track at the loading and dumping stations. On going from double track to single track, each train throws the switch by running through it; the train then comes out on to the same line used going in. The Granby cars of one train are dumped at the lower end of the incline while the other train is being loaded.

The main bin at the lower end holds about 150 tons. By means of a Ross chain feeder, the waste rock is delivered to a 36-in., 8-ply belt conveyor, operating on a grade of 7 per cent for 800 ft., at which distance from the bin it discharges into a distributing box. From this point three belts, radiating with an angle of about 30 deg. between the two outside belts, each terminating in a stacker, discharge the waste over the head pulley into the water. The stacker is lightly constructed of timber, with the head pulley overhanging the truck, so that the dump will be built up far enough ahead to allow an advance of 20 ft. at a time. Conveyor drive mechanisms are in the junction house. The main belt conveyor may be controlled from either end. In changing the direction of flow of the rock at the junction, some elevation is lost, which is regained by advancing the stacker conveyor on a grade of about 8 per cent. This grade is desirable also on account of the nearly constant settling that takes place on the outer edge of the dump. Coarse waste disposal is performed on two shifts during the 24 hours. The average total operating cost per ton of waste from stockpile to the dump is 5c., which includes expense of new construction necessary for the expansion of the dump.

Recently a second unit, consisting of bin and stacker conveyors, has been established 2,000 ft. south of the present bin. The incline tram will not deliver cars to this point by gravity, so a 9-ton electric locomotive handles the train from the bottom of the present incline to the new bin, after which the operation is the same as in the first unit. Waste rock is used extensively in and around Juneau for various purposes, and bins for the various sizes have been provided near the main bin at the lower end of the incline tram. Graded waste rock is used for filling the Juneau waterfront, for general road ballast, sub-surface and finishing surface, concrete aggregate, and railroad ballast.

Since 1920, when coarse waste was first disposed of in this manner, more than 20,000,000 tons of rock has been dumped into Gastineau Channel. The rock pile extends into the channel about 1,300 ft. from the shore line, and reaches an elevation of 75 ft. above high tide. In area above the water line it covers more than 20 acres. Inasmuch as the prevailing wind at Juneau sweeps up the channel from the south, the coarse waste dump affords great protection to the water front.

During the first years of dumping operations the outer end of the rock pile settled, and even broke off and slid out of sight several times. After this experience the steel stackers were replaced with portable wooden structures: the conveyor frame itself, with plenty of timber, was built in 20-ft. sections. In subsequent slides the timber structure kept afloat, thereby saving nearly all the equipment. At this period an extra stacker was usually provided close to the junction house, so that in these emergencies no time was lost. This sliding and settling of the dump generally coincides with the periods of extreme low tides. In Juneau, the spring and fall tides range from a 20-ft. high tide to a minus 4-ft. low. Soundings in the channel, before dumping was started, showed depths of from

100 to 120 ft. with a flat muddy bottom. The bedrock probably forms the sides of a U-shaped valley, the bottom of which, to within 100 ft. of the top of the water, is filled with fine material from near-by creeks. Lately, the weight of the rock pile has forced the mud bottom up in front of the rock pile, making a shoal for 200 ft. or so beyond the end of the dump. For this reason a limit for the dumping of coarse waste has been set at 1 500 ft. from the shore line. The shoals that have appeared from time to time are rapidly worn away by the strong tidal current. The channel is wide and navigation is not menaced.

The total operating cost per ton rejected, for crushing, sorting, and disposing of waste rock, is presented in the accompanying table.

	1928	1929	1930	1931
Coarse crushing per ton discarded	$0.0333	$0.0313	$0.0305	$0.0262
Sorting	.0394	.0339	.0368	.0327
Coarse screening	.0015	.0013	.0010	.0006
Waste rock disposal	.0432	.0551	.0372	.0655
General operation and maintenance	.0182	.0183	.0177	.0171
Total	$0.1356	$0.1399	$0.1232	$0.1421
Total tons discarded	1,922,949	1,815,969	1,854,571	1,863,332
Total cost of fine milling per ton fine milled, for comparison with the above figures	$0.3264	$0.3145	$0.3207	$0.2760

The difference of cost in favor of sorting is apparent; however, when the hardness and absence of fine material in the waste is considered, to crush, grind, and concentrate what is now discarded could not possibly be done at the prevailing cost for milling. This, and in view of the low gold content of the rejected waste, makes sorting an important step in the beneficiation of Alaska Juneau ore, permitting economical and efficient operations.

Safety, Welfare, and Labor Compensation

Success of safety and welfare measures depends to a large extent on the degree of interest and concern manifested by the directing officials of the company, and on its operations. At Alaska Juneau the problem of insuring a creditable record in the mine is the direct responsibility of the chief engineer; a permanent committee —consisting of the mine superintendent, foremen, timber boss, shift bosses, and safety engineers—is possessed of the necessary technical knowledge of changing conditions underground to anticipate accidents and minimize their occurrence. Education of employees in safety-first methods and practices has proved highly successful; 80 per cent hold first-aid certificates issued by the U. S. Bureau of Mines. The company provides complete medical service and hospital facilities for all on the payroll and their families, at a cost to each employee of only $2 per month. Accommodation and board for single men is available at cost. A feature of employment practice is the payment of bonuses, for all employees, regardless of rate of pay, which has added to the stabilization of labor supply. The annual sum thus expended by the company amounts to about $50,000. The system affords an excellent example of the means by which a company can create a personal interest throughout the organization in the advantage of increased production and economical operation. Contract work is a feature of Alaska Juneau mine practice.

L. H. Metzgar
General Superintendent

The hazards of mining are perhaps slightly more at the Alaska Juneau than in the average mine. The ground stands well, and accidents caused by falls of rock from the back are very infrequent. Owing to the coarse nature of the caved material handled through drawholes and grizzlies and loaded into the cars, and to the large tonnage handled, bulldozing and loading introduce more hazards than in mines where the material handled is finer, and where tons per man per day are fewer.

Because the cars and locomotives are large in proportion to the mine opening, and for the reason that long trains are handled, the hazard that attends the general operation of tramming is greater than that in mines using lighter equipment and smaller trains.

Every accident that results in an injury necessitating a first-aid dressing is reported to the office of the company on a regular form, whether or not the injury results in any lost time. The company physician and surgeon keeps a record of all men calling at his office for dressings or on account of sickness. Each evening he mails a brief report relative to employees that called at his office and those sent to the hospital during the day. A monthly hospital report is made showing the number of cases, the nature of each ailment or injury, and the progress each case has made. If an employee is sent to the hospital on account of injury, the doctor immediately makes out a preliminary report that shows the nature and extent of the injury. For a severe injury, a progress report is made from time to time until the case is discharged, at which time a final report is made. All reports for each case are filed together, so that a complete history of the case is available in convenient form. A record of the surface and underground accident rates per 10,000 shifts is shown in the accompanying illustration.

Accidents that involve lost time are reported monthly to the Territorial mine inspector. Compensation is paid in accordance with the Territorial Compensation Law, the company taking care of the adjustment of all payments of compensation for its own account. Payment of compensation is made directly to the employee to whom compensation is due, and as soon as the extent of the disability to be compensated for is determined. When the temporary disability of a workman is of considerable duration, payment on account is made to the employee from time to time if he so desires. In nearly all cases the representative of the company and the employee are able to agree on the amount of compensation to be paid, and no intermediary is required.

A surface safety committee, composed partly of members elected by the employees and partly of members appointed by the company, meets regularly and reports on all accidents occurring on the surface, in the shops, and in the mill. This committee, which generally includes some of the foremen among its members, is free to investigate any serious accident, and to inspect any defective equipment or improper working condition. The committee, in addition to reporting on accidents, makes recommendations for safe practice and for improvement to facilities and equipment.

Safety in the mine is organized differently, in that it is directly under the chief engineer, and the committee is composed of the mine superintendent, mine foremen, timber boss, three shift bosses, and two safety engineers. Because working conditions in the mine change rapidly at times, and as labor turnover is greater than on the surface,

this organization has been found to function better in the mine than an elective committee.

The duty of the safety engineers is to instruct new employees in safe practices, and to assist in seeing that proper facilities for safe working are provided. All reasonable recommendations pertaining to safe practices that may be made by either safety committee, individual members of the safety committees, or by any employee are tried, and, if found practicable, they are adopted.

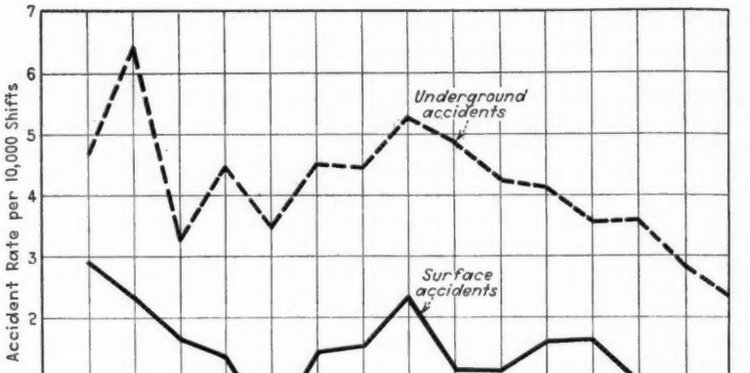
As the result of a campaign in first-aid training last year, 80 per cent of the employees now hold first-aid certificates from the U. S. Bureau of Mines. The management believes that by first-aid instruction and training a better appreciation of the necessity of safe practice will result.

No welfare work is undertaken except that which pertains to medical care and hospitalization of the injured and sick. The laws of Alaska authorize an employer to deduct $2.50 per month from each man's pay as medical and hospital dues. For this the employer is required to furnish medical care and hospitalization for an injured employee for a period not to exceed one year.

Alaska Juneau deducts $2 per month from each man's pay for medical care and hospitalization. The amount so collected is all expended in upkeep of the hospital and in salaries to its staff. The employee receives, in addition to what is required by law, medical care and hospitalization in the event of sickness. Employees and their families also receive free medical service at the office of the company physician during his regular office hours.

During earlier operations through Gold Creek tunnel a dormitory and mess house were constructed near the portal of the tunnel to accommodate 90 men. This dormitory and mess hall have been used continuously ever since, and they are now used to accommodate new mine employees during the earlier periods of their employment. The mess hall is so conducted that the price for board, on the average, meets the operating cost. No profit is made. Inasmuch as all employees have an opportunity to patronize the different business houses in the town of Juneau, no commissary is operated by the company.

Since 1916 nearly all driving of development headings and stope preparatory work have been by contract. When contracting was started notices were posted and bids received for driving certain development headings. Calling for bids on each piece of work is not present practice, because, through long usage, equitable prices have been determined for various kinds of work under prevailing conditions. In fixing prices, the nature of the ground, accessibility, and many other factors peculiar to the work



are considered. From time to time, in the instance of an important piece of work, notices are posted and bids received. This gives new men not in the employ of the company an opportunity to bid, and tends to keep contract prices in accordance with labor conditions and labor supply. A representative scale of contract prices appears in Table I.

Table I—Scale of Contract Prices

Work	Dimensions, Feet	Price per Lineal Foot
Main level openings...	9x9	$12.50–$14.50
Drifts and crosscuts...	7x5	7.00– 8.50
Powder drifts.....	4x3	5.50– 6.50
Stope raises.....	5x5	6.50– 8.50
Ore transfer raises.....	7x7	10.00– 12.00

Under these contracts explosives for breaking the ground and all labor, including that for removing broken muck, are furnished by the contractor. Explosives are charged at a fixed price, regardless of market fluctuations. The company supplies all tools and compressed air and lays track and pipe. Drills are kept in first-class condition, and plenty of sharp steel is available. Air at 90-lb. pressure at the drill is supplied. When a contractor is doing well, a small amount of his earnings are generally retained to insure satisfactory completion of the work. In a few instances the contractor has forfeited this retained pay. A recent development of the contract system has been to limit the contractor's earnings to a certain amount, and to divide the balance earned among the contractor and the contractor's employees, on the basis of the number of shifts worked. A typical contract statement is presented in Table II.

Both the contractor and the men employed by the contractor are carried on the company payroll and paid by check. The contractor receives a minimum base wage and the contractor's employees are paid a certain base wage whether or not the contractor earns it. In the event that the contractor does not earn the base wage, the deficit is carried as a charge against the contract and deducted from future earnings of the contract. If indications are that the contractor cannot earn enough to make base wages, or more, a new contract is let. If failure to make base wages results from lack of skill on the part of the contractor, a new contractor is employed. If failure is caused by conditions over which the contractor has no control, the rate is changed.

Contracts are kept in small units. Where more than one shift is worked, a partnership contract is let, with one contractor on each shift. The contractor seldom has more than four to six men in his charge. Although the contract system requires less supervision than company work, the contractors are nevertheless all closely supervised by the mine superintendent, foremen, and the engineering department. Progress of the headings and rates for contract work are carefully watched by the engineering department, and the rates are revised from time to time if circumstances warrant such action.

Bulldozing and tramming were originally on a contract basis, the contractors doing the bulldozing and subletting the tramming. This work is still carried in the company contract accounts, but several conditions have so modified it that it is now more of a bonus than a contract arrangement. Under this arrangement a certain rate per ton is

Table II—Typical Contract Statement

Contract gross earnings: $2,750

Contractors' employees, 156½ shifts @ $5.50	$860.75
Liability insurance	41.00
Powder, 49 cases @ $9.50	465.50
Primers, 950 @ $0.08	76.00
Contractors: 31 shifts 31 shifts 30 shifts	
92 shifts @ $9	828.00
	$2,271.25
Balance for bonus	478.75
Total	$2,750.00

stipulated for bulldozing and tramming, the contractors paying for all explosives and furnishing the labor for loading and tramming. The bulldozing bosses, or men who supervise and assist in the bulldozing, are carried as the contractors, and are guaranteed $1.50 per day more than the base rate for bulldozers, which at present is $6. All the men on this contract are guaranteed a certain base wage, and this wage is paid whether the contract earns it or not. In the event that more than the base wage is made, the surplus is prorated among the 150 men working on this contract, in proportion to the base wage. If less than base rate wage is made, the deficit is carried forward and deducted when earnings exceed the base rate wage. A typical bulldozing contract statement is presented in Table III.

Table III—Typical Bulldozing Contract Statement

		Dr.	Cr.
356,900 tons @ $0.15			$53,535.00
Explosives		$18,888.40	
103 shifts, pay of men who left		569.00	
Continued from contract 58-A		2,558.68	
Labor at base rates:			
2,261½ bulldozers @ $6.00	$13,567.50		
268 shifts chute punchers @ $6.25	1,675.00		
360 shifts, motormen @ $5.00	1,800.00		
484 shifts, loaders @ $5.00	2,420.00		
508½ shifts, clean-up men @ $4.75	2,414.18		
444 shifts, contractors @ $7.50	3,330.00	25,206.68	
Liability insurance		850.03	
		$48,072.79	48,072.79
Available for bonus			$5,462.21

	Bonus Rate	Bonus	Bonus Plus Rate
444 shifts, contractors	$2.00	$1.58	$9.08
2,261½ shifts, bulldozers	1.75	1.38	7.38
268 shifts, chute punchers	1.75	1.38	7.63
360 shifts, motormen	0.75	0.59	5.59
484 shifts, loaders	0.75	0.59	5.59
508½ shifts, clean-up men	0.50	0.39	5.14
116 shifts, timbermen	1.75	1.38	7.38–6.88
137 shifts, timbermen helpers	1.00	0.79	5.79–5.29
30 shifts, electricians	0.75	0.59	6.59
477½ shifts, continued from contract 58-A	0.75	0.59	5.84–5.59 5.34–4.84

In March, 1924, because so much of the mine work was on a contract or bonus basis, an endeavor was made to create a bonus arrangement whereby all employees would share in any increased efficiency that might be effected. The following extracts are taken from a notice then posted that outlined the features of this proposal:

"With the idea of creating interest in production and a desire to share with its employees the advantage gained by an increased production, the Alaska Juneau Gold Mining Company proposes to put into effect a bonus system.

"It should be understood that this is in the nature of an experiment, that the terms are subject to readjustment when or if conditions make it necessary or advisable. It is done by the company in good faith and in an earnest desire to better the conditions of employees so far as it is possible to do so; sincerely hoping and believing that the employees will accept the bonus with equal faith and that the result will be mutually beneficial.

"Beginning as of March 1 a bonus will be paid to all employees of the Alaska Juneau Gold Mining Company in the mill and beach division as follows:

"This bonus will be computed on the shifts worked, based upon tons per man per day, and will consist of an amount equal to one-half of the labor cost on all tons over 30 tons per man per day.

"The provisions of this distribution are that a man to participate in the bonus must have worked 26 shifts and be working at the time the distribution is made. The bonus distribution will be made on the twentieth of each month.

"No bonus will be paid on overtime. In the event a man is incapacitated by accident, he will be paid bonus on such shifts as he may have worked during the month in which he was incapacitated by accident.

"The company has spent and is spending large sums of money to improve operating conditions, increase output, and reduce operating costs. In introducing this bonus the company is sharing with the employees the advantage gained by such expenditures in order to increase the interest of the employees in their work and share with them any saving in cost that can be effected by improving working conditions and a further increased efficiency of labor."

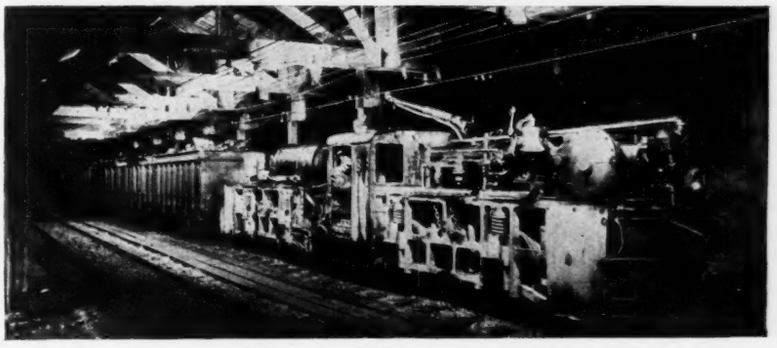
As may be noted, this bonus was paid at a different time and by separate check, the regular wage being paid on the tenth of the month. The arrangement was later expanded to include all employees who were not working directly on contract work.

The bonus arrangement was put into effect in the manner outlined in the notice, and at a time when labor was somewhat scarce. This arrangement is based on the theory that during a time of labor shortage, repairs and improvements may be neglected, and the employees available may be devoted to production. If the tonnage is kept up with a smaller total number of employees, tons per man per day increase, and wages are automatically increased. When labor again becomes more plentiful, the number of employees is increased, and repairs and improvement work are resumed, thereby automatically decreasing wages.

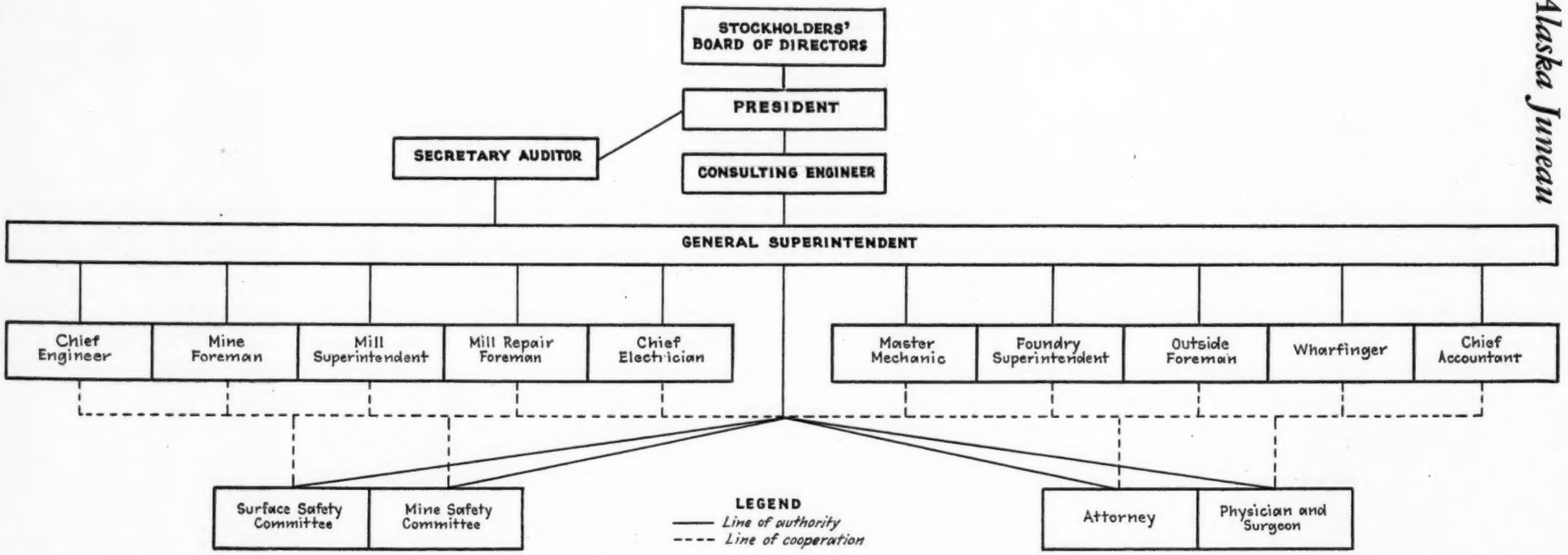
The bonus arrangement has been modified from time to time to meet conditions, and has been found useful in stabilizing labor supply. The bonus is paid in equal amounts to all employees, regardless of their rate of pay. During the period of high prices it was especially helpful to employees in the lower part of the wage scale. From the time the bonus arrangement was made effective until Dec. 31, 1931, about $50,000 per year was distributed to employees other than those working on contracts.

For a number of years the work of the mine crew was supervised by a foreman, timber boss, three shift bosses, and a tramming boss. The fact that all development work was on contract to seasoned and experienced miners made such a simple organization possible. As the number of working faces and contracts increased and new contractors were employed, the chief engineer was given supervision of contract work. He, in close cooperation with the mine superintendent, who has direct supervision of the work, is responsible for the letting of the contracts and the satisfactory performance of the work.

Recently, owing to the fact that deep development operations have been added to the development program, the mine organization has been increased, so that present personnel consists of a mine superintendent, a mine foreman, a timber boss, a night foreman, three shift bosses, a tramming boss, and two safety engineers. All development headings still remain on a contract basis, but the increased amount of work and the greater area over which it is distributed require a larger staff for its proper supervision.



Organization Chart



CHIEF ENGINEER———*Responsible for:* Engineering work in all departments; mine surveying; mine sampling; direction and layout of development; measurement of mine contract work; determination of contract prices; preparation of mining and milling records and other data for monthly reports; drafting; cost estimates.

MINE FOREMAN———*Responsible (subject to direction of chief engineer) for:* Supervision of mining operations; development; stoping; bulldozing; loading; drill steel sharpening; drill repairs; timbering and chute building; underground pipe and track work; mine labor.

MILL SUPERINTENDENT———*Responsible for:* Supervision of milling operations; coarse crushing; sorting; fine crushing; concentration; retreatment; amalgamation; bullion refining; mill sampling; experimental metallurgical tests; daily delivery of amalgam to main office vaults; trimonthly collecton and transportation of bullion to refinery; return of refined bullion to office vaults; tri-monthly transportation of concentrates to wharf.

MILL REPAIR FOREMAN—*Responsible for:* Repairs and replacements of all mill equipment; service records of all types of mill equipment; repair and maintenance of mill buildings; arranging and maintaining repair schedules so that mill operations are delayed a minimum period.

CHIEF ELECTRICIAN ——— *Responsible for:* Supervision of all electrical projects; operation and maintenance of hydro-electric power plants and steam power plant; repair and maintenance of power transmission lines; repair and maintenance of mine and beach locomotives; repair and maintenance of all motors and other electrical equipment in all departments; keeping service and repair records for all electrical equipment; maintenance of adequate stocks of electrical supplies and replacement parts through cooperation with the supply department.

MASTER MECHANIC———*Responsible for:* Supervision of all mechanical operations, except the foundry and repairs to mill equipment; machine shop and boiler shop work for all departments; construction and repair of ore cars; welding and cutting; repairs to hoisting equipment and tipple; compressor maintenance and repair; outside airline maintenance.

FOUNDRY SUPERINTENDENT———*Responsible for:* Supervision of foundry and pattern shop; castings for all departments; maintenance of adequate stocks of standard castings for all departments; supervision of company property at Treadwell.

OUTSIDE FOREMAN———*Responsible for:* Supervision of all carpentry, except mine timbering and mill carpentry; flume construction and maintenance; construction and repair of snowsheds, buildings, and wharves; maintenance and operation of waste disposal system.

WHARFINGER ——— *Responsible for:* Supervision of wharf and supply department; receipt, storage, and recording of all supplies shipped to the company; filling of requisitions issued by all departments; delivery of requisitioned supplies; receipt and shipment of concentrates, as well as other outgoing materials; maintenance of stock records and inventories; renewal of depleted stocks.

CHIEF ACCOUNTANT———*Responsible for:* Supervision of all accounting at Juneau; timekeeping for all departments; payroll for all departments; payment of all bills against the company; keeping of cost accounts; preparation of monthly reports on operating costs and other financial phases of operations; stenographic work; daily receipt of amalgam output from the mill and its storage; tri-monthly delivery of amalgam to the refinery; receipt and shipment of refined bullion; ordering and distribution of all printed forms required by the various departments.

SURFACE SAFETY COMMITTEE———*Responsible for:* General supervision of surface safety; investigation and study of all surface accidents; suggestions of methods of accident prevention; suggestion of safety rules to govern surface operations; organization and maintenance of surface first-aid crews.

MINE SAFETY COMMITTEE———*Responsible for:* General supervision of mine safety; investigation and study of all mine accidents; suggestion of methods of accident prevention; suggestion of safety rules to govern mine operations; organization and maintenance of mine first-aid crews.

ATTORNEY ——— *Responsible for:* Supervision of all legal phases of the company's operations; recording and filing of all deeds, contracts, affidavits of assessment work, and patent applications; the conduct of all lawsuits of the company; interpretation of legal instruments for the company.

PHYSICIAN AND SURGEON———*Responsible for:* Supervision of all medical phases of the company's operations; medical and surgical treatment of all injured employees; medical and surgical treatment of employees and their families in ordinary illness; first-aid instruction of employees; supervison of hospital.

In view of the recent history of Alaska Juneau, Mr. Wernecke's opinion as to what may reasonably be expected in the way of future ore development is of interest. Avoiding a discussion of broad geologic conditions, he draws upon data accumulated by the company and upon several general studies of the area, to present the results of closer investigations of the ore zones and to show their relation to mine work. He pays particular attention to the north orebody, which promises to be of great importance in the future. The irregular mass of slate outcropping on certain claims of the Ebner group, together with the included fingers of metagabbro, he says, will undoubtedly include most of the ore in the mine. Where slate and gabbro fingers are interwoven below a certain chonolith, unusually good ore should be found. A continuation of this mass of slate contains the bulk of the ore in the south orebody which has been so profitable. . . . Discarding other theories, Mr. Wernecke cites as most probable three possible origins of the quartz veins formed in the biotized rocks. Growth of the veins he attributes to repeated opening on their walls, followed by new precipitation. He explains the irregular distribution of the veins and of the gold in and on them. The gold is younger than the quartz, in his opinion.

Livingston Wernecke
Consulting Geologist

Geology of the Ore Zones

Broad geological conditions of the Juneau gold area have been described by A. C. Spencer, F. B. Hyder, H. M. Eakin, and J. B. Mertie, Jr., one of the earliest accounts being that of A. C. Spencer. Information presented here concerning the several crustal oscillations is derived from publications of the U. S. Geological Survey and the Canadian Geological Survey; in particular, U.S.G.S. Bulletin No. 287, by A. C. Spencer; No. 800 by Buddington and Chapin; and bulletins by J. B. Mertie, Jr.; and Memoirs 31 and 51 by D. D. Cairnes, and Memoir 150 by W. E. Cockfield and A. H. Bell, of the Fourth Canadian Survey. Early geological reports on the Alaska Juneau were also made by Oscar Hershey, Fred Searles, Jr., and Andrew Lawson. Hershey determined the rock nomenclature in 1913 that was used at the Alaska Juneau, although Spencer had established a tentative one in 1906 in his Bulletin No. 287. Detailed studies of the geological character of the gold zones in the mine have been made but have not been published. The object of this paper is to present the results of closer studies of the ore zones and to show their relation to mine operations.

Inland, and forming the core of the Coast Range, is a great diorite batholith. Between it and the coast are more or less parallel belts of schists, slates, and igneous rocks that dip at high angles of inclination inland at some distance from the westerly diorite contact. The series of enveloping rocks is believed to represent folding and crumpling accompanying mountain building and batholitic intrusion. Erosion has removed great masses of these enveloping formations and has exposed the batholite core in places.

The oldest rocks of the Juneau area are schists and slates. The slates have been assigned to Early Mesozoic or Triassic by Spencer, Eakin, and Martin, but the evidence of this age is not convincing. Originally they were muds and silts, and their deposition was followed by volcanism in the Late Triassic, which formed beds of tuff and flows of andesite. This series of rocks now outcrops along Gold Creek, west of the Alaska Juneau mine. Although some beds of slates have been found intercalated in the tuffs, no tuffs or andesites have been mapped in the slates, but possibly some do exist and have been mistaken for amphibolitic gabbros, so completely has subsequent metamorphism changed all igneous rocks to amphibolitic schist. The greenstones of tuffs and lavas vary from a dense grayish-green fine-grained andesite to a dark-green fine acicular-grained amphibolitic schist. The denser andesite, composed of actinolite, chlorite, and a recrystallized ground mass of quartz, calcite, and some albite, is almost identical in appearance and composition with the amphibolite schist derived from a hornblende gabbro. Thus, to distinguish the two types is impossible, unless transitions of the andesite to amphibolite are found in the field. The similarity also may have caused some errors in mapping areal geology, particularly near the slate greenstone contact. Apparently, the transition from the principal body of slates to tuffs and andesites is in most places conformable, but the exposure of this contact zone in the Ebner adit suggests an unconformity. An alteration of bands, 10 to 25 ft. wide, of thin-bedded, light- to dark-gray quartzites and flaggy slates, with beds of tuff and andesite, 15 to 100 ft. thick, marks the transition. Some evidence indicates that they are members of the greenstone series, the same as the slates near the center of the greenstone, and not members of the slate series.

The structure of the slates and greenstones is probably a series of several closed folds that have been overturned so that all bedding dips northeastward. Infolding of the Triassic greenstone accounts for various patches of greenstone. Subsequent intrusions of gabbro, development of slaty cleavage, and accompanying metamorphic changes have obliterated most of the evidence used in correlating different stratigraphic horizons. Consequently, no detailed structures can be determined, and some doubt exists concerning the exact position of the slate-greenstone contact. If the lavas, tuffs, and slate are of the same age, a transition zone would be expected between the two, but if the greenstones are Triassic and the slates Carboniferous, no transition can be expected, but, instead, an unconformity.

Following the deposition of the Lower Jurassic and Triassic volcanics, another series of slates and volcanics was formed, which now outcrops on Douglas Island. They consist of black slates overlain by volcanic tuffs and agglomerates, and have also been assigned to the Early Jurassic period by Spencer and other geologists. A reasonable supposition is that during this period of volcanism the sills and laccoliths of gabbro (now metagabbro) near the Alaska Juneau were intruded under deep-seated conditions into the earlier Mesozoic (or late Paleozoic) tuffs and slates.

The metagabbro, originally a hornblende gabbro, was intruded as irregular fingered laccolithic bodies, or chonoliths, and long, narrow sills 10 to 200 ft. in width. When intruded in the greenstones where both have been altered to amphibolite schist, the metagabbro is difficult to identify. It varies in appearance from dense dark-green blasto-dioritic rock, specked with light gray spots of secondary minerals from the original feldspar, to a dark-green fibrous amphibolite schist, with little microscopical evidence of the original feldspar contents. For such small intrusive bodies, the contact aureole in the slates is remarkably prominent, being a margin 10 to 100 ft. wide of spotted schist surrounding the intrusives. The center of the spots usually contains calcite, but occasionally

garnet. Such spotted schists are formed at great depths under conditions of rock flowage, where high temperatures and pressures prevail, and where some water was present. Later, these schists, with other surrounding rocks, were altered by pneumatolytic and hydrothermal agents, which made the spots more prominent by adding biotite to their margins.

The first members of the Coast Range batholith were intruded into the Alaska Juneau slates, Triassic and Jurassic greenstones, and Jurassic gabbroic sills and chonoliths near the close of the Jurassic period. The prevailing rock type of the batholith is a gneissoid granodiorite, part of the gneissic structure having been developed during the consolidation of the granodiorite, and part after consolidation, by orogenic pressure. A tongue of gneissic granodiorite, 500 to 1,000 ft. in width, outcrops in Gold Creek below the old placer pits and extends up the mountain on the north side to the summit of the ridge.

Satellitic dikes of aplite and quartz accompany the granodiorite. The quartz dikes that may have accompanied the gneissic granodiorite have not been distinguished from those of later age, and consequently are included with the younger group in mapping and in descriptions.

Several other granitic invasions into the surrounding rocks followed the gneissic diorite, and by the end of the Jurassic or Middle Cretaceous the Coast Range batholith was formed in outline as it is today. These later granitic instrusions are represented in the Juneau district by the albite-diorite dikes at Treadwell and, probably, by additional quartz veins, which are auriferous at the Alaska Juneau and adjoining mines. These later veins and the associated biotization resulted from the escaped volatile constituents of some granitic mass at depth, and are not directly related to the nearly gneissic granodiorite tongue mentioned before, because a long time interval elapsed between the solidification of the gneiss and the formation of the auriferous quartz veins.

The gneissic texture, regional schistosity, slaty cleavage, and recrystallization of the gabbro to metagabbro, and of andesite to meta-andesite, were formed by metamorphism that resulted from a regional uplift preceding and accompanying the later granitic masses. As the later auriferous quartz veins and associated biotite rocks are not characterized by corresponding textures, they were formed after the diastrophic forces had subsided, possibly in the Middle or Late Cretaceous. Earlier quartz veins, however, existed prior to the diastrophism, but not much study has been given them, as they are barren.

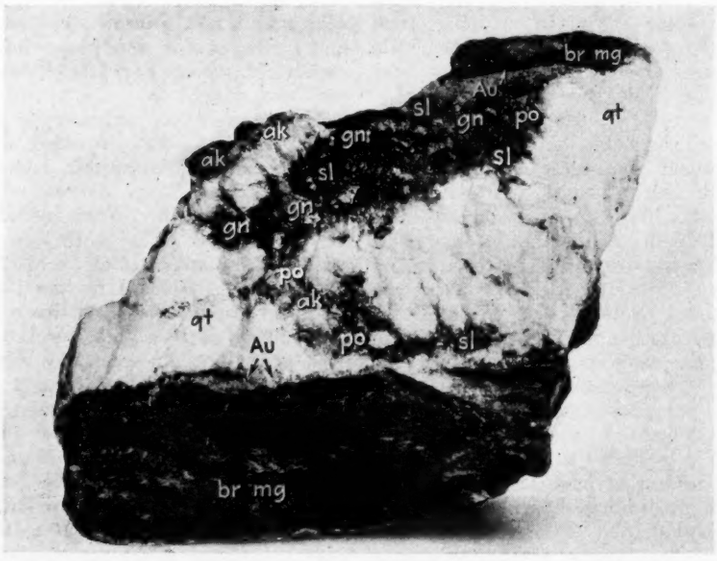
The more volatile emanations of some member of the Coast Range batholith penetrated a zone 1,000 to 2,000 ft. in width near the contact of the greenstones and slates, where earlier gabbroic intrusions had formed a zone of weakness, intensely altering the surrounding rocks and introducing numerous quartz stringers, many of which are in bands that form stringer lodes. The mineralized zone that has been profitably productive in a few places, notably on the Alaska Juneau property, has been traced for 3½ miles.

In its earlier stages the mineralization consisted of emanations of water, silica, sodium, calcium, carbon dioxide, iron, sulphur, and small amounts of valuable metals, such as gold, silver, lead, zinc, and copper. The first effects were to permeate thoroughly the surrounding metagabbro, slates, and meta-andesites with silica, light-brown biotite, albite, calcium carbonate, and pyrrhotite, and to change completedy their mineral composition and texture. I have not seen anywhere else such complete and extensive changes in rock textures as has been accomplished by the biotization about the Alaska Juneau. The metagabbro in some instances represents a change from hornblende gabbro to an amphibolitic rock; from a green amphibolitic rock to a brown biotite rock; and from a brown biotite rock to a light-gray carbonate rock.

Pneumatolytic alteration of the metagabbro is represented at different places by three differently appearing rocks; most prominently by a dense fine-grained sugary brown rock, without trace of gabbroic or schistose structure, but containing a small quantity of minute grains of gold widely disseminated and detectable only by assaying. A fine-grained grayish-green conchoidal fracturing phase is next in abundance, containing little gold, and a light-gray siliceous fine-grained rock, with an irregular fracture, is the least common phase. Gold is present in the last phase in small quantities, similar to the brown phase. This phase with quartz veins is the best indicator of an ore band.

The quartz veins, formed in connection with or immediately following biotization, have occasionally tubular shapes and smooth walls when in metagabbro, but usually they are extremely irregular in outline and distribution when in slate. The three most probable origins of the quartz veins are as follows: first, the veins might have originated from concentrated aqueous solutions of silica finding their way under enormous pressure into the slates and brown gabbros, and crystallizing out of solution in a comparatively short interval, and also replacing much of the wall rock, a theory similar to Spurr's veindike theory; second, formation by the force of crystallization from dilute aqueous solutions after entering small fractures in the country rock, a theory similar to Stillwell's explanation of the Bendigo quartz veins; and, third, a theory similar to Adolph Knopf's for the quartz veins of the California Mother Lode region, which involves the leaching of quartz from the surrounding rocks by meteoric waters and its deposition in near-by fractures.

Repeated openings of the rocks on the walls of the quartz veins by the elastic rebound theory for faults have allowed them to grow by new precipitation of quartz in the newer fractures. The gold and sulphides are believed to



Gold-quartz ore. Mag. 0.6. Qt, quartz; sl, sphalerite; gn, galena; po, pyrrhotite; ak, ankerite (with some white mica); Au, gold; br mg, brown metagabbro.

have come from a magmatic source near the close of the quartz deposition, and to have been precipitated on the margins of the quartz veins by mixing of the solutions. How Knopf timed his gold solutions to arrive on the spot where the quartz veins attained their full growth by independent lateral secretion is not explained. For Alaska Juneau conditions, the obtaining of the silica from the adjoining rocks is requiring too much of the basic gabbros and clay slates. They are now charged with silica from replacement by magmatic solutions that also formed the quartz veins along fractures, but originally were deficient in silica.

In my opinion, neither the theory that the quartz veins were formed by lateral secretion from surrounding rocks and coincident deposition of gold from a magmatic source, when they obtained their maximum growth, nor the theory that the quartz was from a magmatic source and has been deposited in open fissures that were being continually replaced on the margins until they obtained their maximum size at the end of an hydrothermal period, applies to Alaska Juneau conditions.

Of the other two theories, the first has much supporting evidence, and probably after the bulk of the quartz had been crystallized from the concentrated solutions, dilute solutions of silica were circulating during hydrothermal conditions and deposited the gold with ankerite and pyrite in the outside of the quartz bodies and in the banded quartz along the footwall fault.

Irregular distribution of the quartz veins in the mineralized zone is primarily caused by the irregular fracturing resulting from the variation in the physical properties of the surrounding rocks. The shape of the irregular

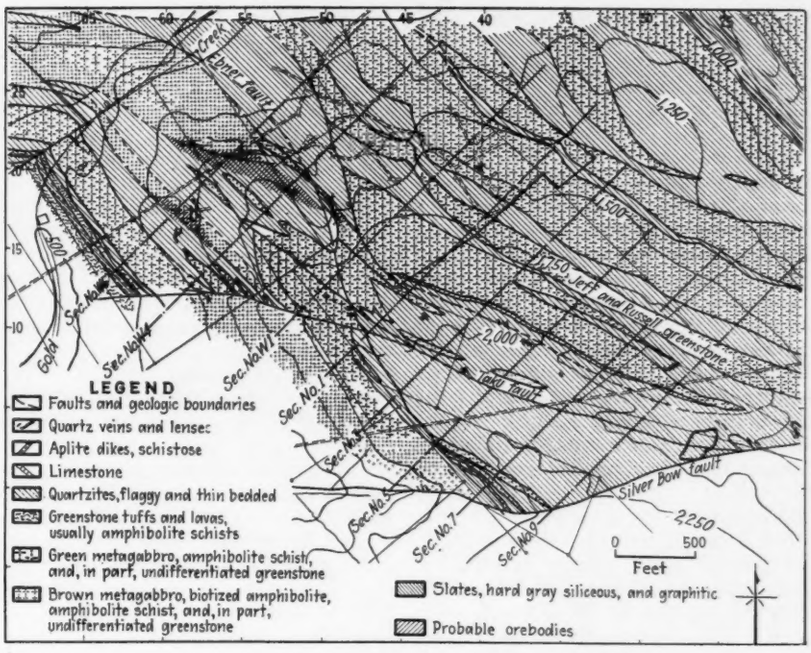


Fig. 1—Geologic surface map of the north orebody

fractures in the slate suggests that they may be attributed to the tensional or torsional stresses such as may accompany a local uplift.

The quartz bodies are arranged in the broad mineralized zone as irregular, elliptical, isolated, or disconnected groups of stringers that are more or less continuous bands of varying widths and in more or less continuous pipe-like groups along fracture ridges or tongues of brown gabbros. Isolated or disconnected groups of stringers that are found occasionally in the center and near the hanging wall of the ore zone have had a shorter or interrupted connection with the source of supply and, hence, less opportunity to develop products of the hydrothermal period, and thus less gold; or, perhaps, they were formed with the earlier barren quartz deposition. The isolated groups are not ore.

The more or less continuous bands of pipe-like groups of quartz stringers, which have much ankerite and sericite, calcite, albite, and sugary quartz, and quartz veinlets developed in the intervening country rock, are most favorable for orebodies. Shoots of high-grade ore ($2 to $2.50 per ton) may be expected in these bands, particularly where the pre-mineral fracturing has been extensive, where quartz bodies are abundant, and silicification is extensive, as at the junction of two or more zones of fracturing and near the points and along the contacts of some tongues of gabbro. However, an occasional section of an ore band with an abundance of quartz stringers has been found that contained, relatively, a small amount of gold. In those instances, ankerite, white mica, galena, sphalerite, and other products of the hydrothermal stage were also absent.

The irregular distribution of the gold in and on the quartz veins is a result of that metal having had ample time and opportunity, while in solution, to collect at a few centers of crystallization, thus forming the coarse flakes, grains, and nuggets.

A remarkable fact is that in the quartz veins of the Alaska Juneau, Bendigo, the Mother Lode of California, and Nova Scotia the gold is younger than the quartz, and is deposited on the outside of the quartz or in fractures in the veins, associated

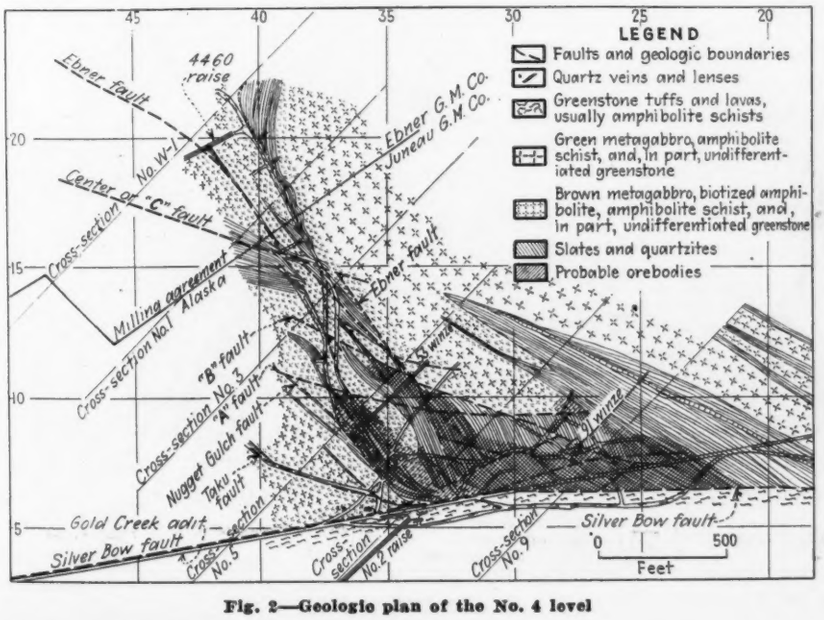


Fig. 2—Geologic plan of the No. 4 level

with muscovite, ankerite, and some pyrite. Like effects should have had like causes, even though different eminent geologists have formulated different theories for the different deposits.

The intrusion of the Coast Range batholith initiated a period of land elevation along the Pacific Coast, but this elevated land mass was eroded to an early mature topography by the end of the Cretaceous and much of it again submerged. The Laramie revolution at the end of the Cretaceous also affected the southeastern Alaskan coast, and at the end of the Eocene or Early Miocene another uplift began, reducing the topography to a peneplain by the end of the Miocene. Minor crustal oscillations brought one of major importance near the end of the Miocene or Early Pliocene, when the general level of the entire Pacific Coast was raised to above the present Coast Range mountains. Later still, during the Pleistocene, the mountains of the St. Elias Range were lifted even higher than the Coast Range, the uplift, except possibly for a short interval of subsidence at the close of the Glacial epoch, continuing to the present. Final events are the recent uplift of the Pacific Coast line. The Tertiary crustal oscillations were accompanied by faulting and fracturing of the rocks and, in places, by volcanism and low-temperature pressure hydrothermal activity.

Geological events since the formation of the quartz veins in and about the Alaska Juneau mine are only partly recorded in those rocks, and then only by sets of faults, associated fracture zones, intrusions of small basic dikes, and a weak hydrothermal alteration that followed the basic dikes. Major faulting is believed to have occurred in the Early Tertiary, and not later than the Miocene, because the small basic dikes are found following the earlier fault planes, but the data are insufficient to determine definitely the age of faulting or intrusion of the basic dikes, other than that both occurred some time in the Tertiary.

The earliest faults in the Alaska Juneau and adjoining properties follow the Nugget Gulch and Ebner ore bands. They are pre-mineral, and much quartz was deposited along the two fault zones and auxiliary fractures. The faults strike N. 40 deg. W. and dip 60 deg. to the northeast. Displacement is believed to be small, and most of it is thought to be probably post-mineral. Three important post-mineral faults are the Ebner, Nugget Gulch, and Silver Bow. The Nugget Gulch fault is in the footwall of the mine workings of the north orebody. Silver Bow fault is the largest fault known, with any accurate knowledge, in the immediate vicinity of the Alaska Juneau mine. The strike varies from N. 75 deg. E. to east and west, and the dip averages 75 deg. to the north.

Several warps have been found in the fault plane, and at each bend a fault of some magnitude branches into the footwall or the hanging wall. The total horizontal displacement is about 1,800 ft., the hanging-wall side having moved toward the west 1,800 ft. with a vertical displacement of 1,750 ft. Movements along this fault shattered the rocks through a zone 50 to 100 ft. wide, the center being marked by two prominent gouges 6 to 10 ft. apart, and the margins by several subparallel gouges, 10 to 60 ft. distant from the center. Between the two central gouges the rock is extremely shattered; between the several subparallel gouges, it is either coarsely shattered or broken into large blocks by several short gouges, the strike of which is approximately at 45 deg. to that of the Silver Bow fault, and which intersect each other at right angles, the dip being variable. The country rock is thus broken into angle blocks, 100 ft. on an edge.

The relationship of the short faults to the central gouges and marginal faults is illustrated in the geological map of No. 4 level of the north orebody shown in Fig. 2. Subparallel faults on the margins of the Silver Bow fracture have a horizontal strike displacement of 200 to 400 ft. that increased to 1,800 ft. along the main gouges near the center. After the major faulting, the dark-gray diabase dikes were intruded into the principal Silver Bow fault gouges. An interval of weak hydrothermal activity closely followed the diabase dikes.

The Taku fault has had little attention in the past, but may have had

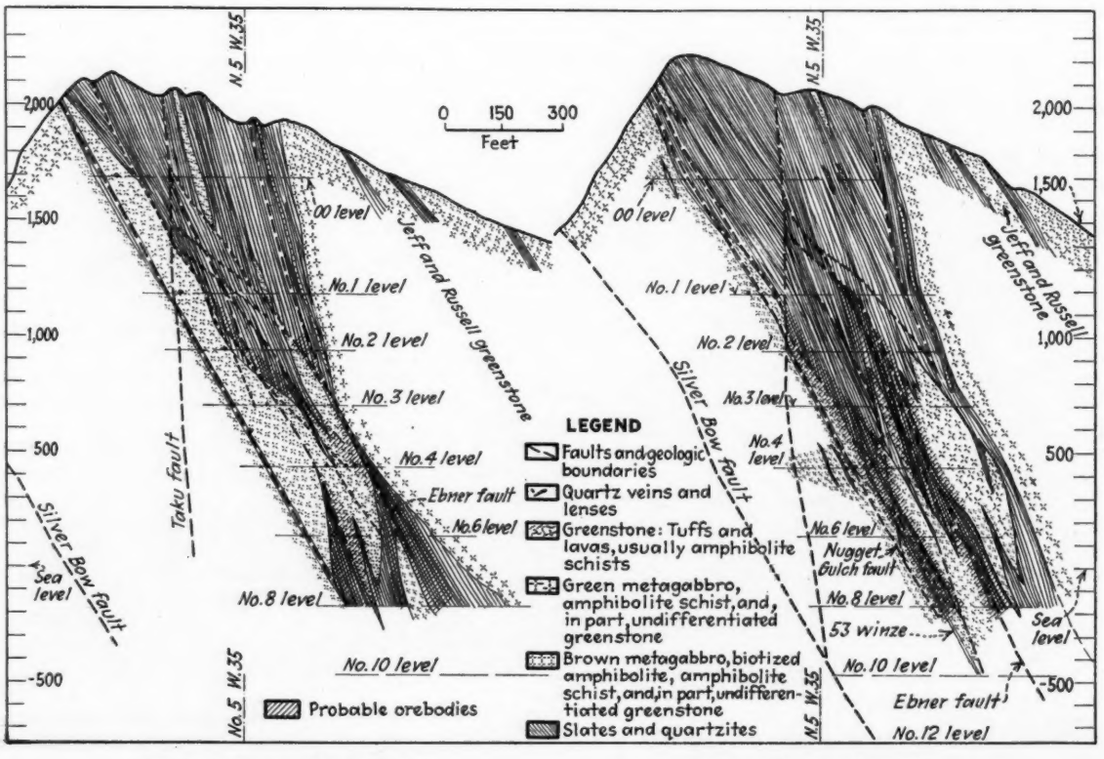


Fig. 3—Cross-section No. 3 **Fig. 4—Cross-section No. 5**

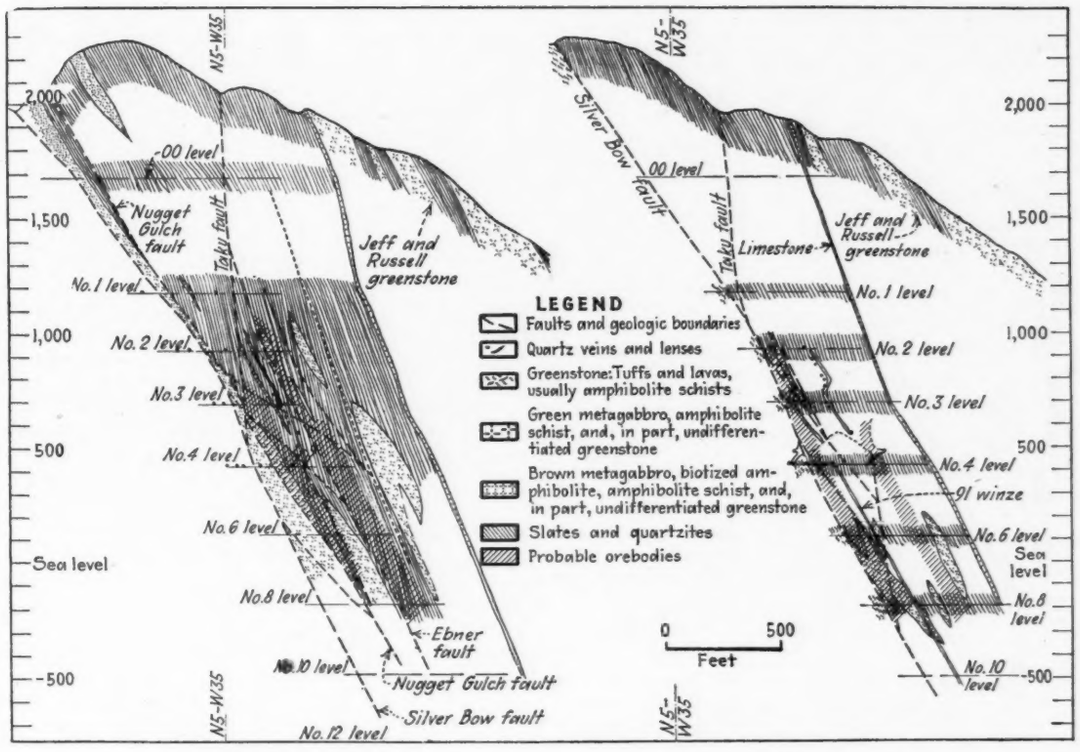


Fig. 5—Cross-section No. 7 **Fig. 6—Cross-section No. 9**

some influence in localizing the large number of quartz stringers and high gold content, where found in the Gold Creek tunnel on the No. 4 level along the Silver Bow fault. Like the Nugget Gulch and the Ebner faults, the Taku fault seems to have been a post-mineral fracture, along which some quartz was deposited. However, the average number of quartz veins and average gold content found to date along this fracture has been too small to make ore. Where it intersects the Nugget Gulch band, and possibly the Ebner band, an unusually large body of ore was formed. On the upper levels, where it is on the hanging-wall side of the Nugget Gulch fault, the Taku fault seems to form the footwall of the ore band, until the ore gradually ends before reaching the surface. Its average strike is N. 70 to 80 deg. W., and the dip is from vertical to 80 deg. north. Other minor fractures and faults have been observed and are contemporaneous with the Silver Bow fracturing.

Silver Bow fault divides the principal ore zone into the north and south orebodies, both of which are being mined. As the north orebody is being developed below the adit on No. 4 level and promises to be of great importance in the future of the Alaska Juneau mine, some of its major geological features are of present interest. A geologic surface map of the north orebody is shown in Fig. 1. Fig. 2 is a geologic map of No. 4 level, the cross-sections indicated being shown in Figs. 3, 4, 5, and 6.

One of the prominent features of the surface of the north orebody is an area of slate, 1,000 ft. wide and 2,500 ft. long, shaped like an ellipse, cut off along a diagonal axis. It is surrounded by green metagabbro, except where intersected by the Silver Bow fault on the southeast; there it adjoins greenstone. Along the margins of this slate are some scattered quartz stringers, the gold content of which is less than normal. The only biotitic mineralization is in a few small scattered bodies of metagabbro within the slate and near the Nugget Gulch fault on the footwall side. Traced down through the several mine levels, the slate is found to be intruded and displaced by much metagabbro. On the No. 4 level, the semi-elliptical area diminishes to one-half the surface area, and a third of the reduced area is metagabbro instead of slate.

In the footwall of the Nugget Gulch fault, which is the footwall of the elliptical body of slate, is a large blocky mass of brown biotized metagabbro or biotized greenstone, the thickness of which has not been determined. It contains a few scattered quartz veins of subnormal gold content, but no orebodies have been found within its mass.

At the northwest side of the Alaska Juneau property, and southeast of the Ebner group, on the Carroll, Hughes, Blaine, Crown Point, and Grand View mining claims, is an irregularly fingered intrusive mass of green metagabbro, having a shape and origin like Daly's chonoliths, the upper side of which has a shape like an irregular plunging syncline and the under side like a compressed anticline. The point or finger of this chonolith, adjoining the footwall of the slates, can be traced downward to the No. 4 level. It has a fairly uniform trend to the east, with a plunge of 350 ft. horizontally for 250 ft. vertically between No. 3 and No. 4 levels; but it is somewhat steeper between the surface and 00 levels. The point of the finger near the hanging wall has a similar trend to that of the footwall finger, but its pitch is more irregular between the surface and No. 1 level. Below that level, and between the No. 3 and No. 4 levels, it flattens noticeably to a slope of 500 ft. horizontally for 250 ft. vertically in an easterly direction. The pitch of the axis of the chonolith is, like that of the footwall finger, uniform between the No. 1 and the No. 4 levels, but it is much steeper and almost directly down the average dip of the surrounding rocks between the surface and the No. 1 level. On the surface the shape of its central part is like a short, flattened hourglass, having a thickness of 200 to 250 ft. at the narrowest place. The narrow section of the hourglass can be traced downward through the respective levels to the No. 4 level and possibly to the No. 6 level. Its average trend is S. 75 deg. E., and the plunge is at the rate

of 100 ft. vertically to 125 ft. horizontally. A clue to the slope of the lower side of this intrusive may be found in the metagabbro contact on the west side of No. 3 level. This body of slate has the shape of those areas on the surface at the contact with the northwestern footwall lobe of the metagabbro, but it has been converted into a schist and is intensely biotized. Possibly this slate is only a large inclusion, but the general appearance at the three places is that of a tongue of the larger mass of the slate below the chonolith. This tongue is sufficiently metallized to be ore.

Fig. 7—Geologic sketch of No. 200 stope in south orebody

The irregular mass of slate that outcrops on the Grand View, Crown Point, and Keystone claims of the Ebner group is beneath the irregular chonolith and will be extremely interesting when developed below the No. 4 level of the Alaska Juneau, because it, with the included fingers of metagabbro, will undoubtedly contain the bulk of the ore in depth in the Alaska Juneau mine. A continuation of this same mass of slate contains the bulk of the ore in the south orebody, which has been so profitable. In the north orebody, the slate mass has been exposed in two small areas on the No. 4 level and in one on the No. 3 level. Consequently, the only place to obtain an idea of its shape, size, and relative position with associated metagabbro and faults is from the outcrop on the Ebner group. On this outcrop the slate is 1,300 ft. wide and 500 to 600 ft. high or thick between the chonolith above and the apex of a large wedge-shaped mass of metagabbro below. In this region above the apex of the wedge of metagabbro, the slate contains many small tongues and lenses of metagabbro, is much crumpled, and is well mineralized. The early mining of the Ebner Gold Mining Company was in this mass of slate. At the elevation of Gold Creek the leg of slate on the hanging-wall side of the metagabbro wedge has a horizontal outcrop width of 450 ft., corresponding to a thickness of 320 ft., and that on the footwall side, 250 ft., corresponding to a thickness of 200 ft.

In general, the foregoing description applies also to the exposures in the south orebody, where the two principal slate bands are separated by a body of metagabbro, as on the No. 4 level, which is geologically 2,000 ft. below the Gold Creek outcrop.

The converging of the chonolith and the apex of the wedge of metagabbro beneath has served to confine the ore-forming solutions to narrow zones, which affords a better opportunity for a more concentrated deposition of gold and other lode minerals. Extraordinarily good ore should be found in the Alaska Juneau in the slate where it is interdigitated with small tongues of metagabbro below the chonolith. The apex of the metagabbro wedge is divided into fingers that are mineralized and gold bearing.

To predict what may be the lower limits of the body of gabbro, and what is below it, is impossible. Two prominent faults accompany the walls of the metagabbro wedge, both of which are associated with quartz bodies and some gold. The one on the hanging wall has been correlated with the Ebner fault at the northeast boundary of the Alaska Juneau mine. At the boundary, and 100 to 150 ft. into the hanging wall of the Ebner fault, is another fault, with which most of the ore and quartz bodies are associated. The Nugget Gulch fault has also been identified in the Ebner adit near the footwall of a sill of brown metagabbro.

Other faults have been closely studied. Both the Nugget Gulch and Ebner faults, which seem to be closely associated with bands of quartz stringers, converge gradually with increasing depth below the No. 4 level, and also converge in strike southeasterly toward the Silver Bow fault. The fact that the ore bands converge argues well for a concentration of gold below the No. 4 level.

The probable extensiveness and intensiveness of the pneumatolytic and hydrothermal alterations below the No. 4 level of the north orebody are, like the structural geology, suggested by the change in the appearance of the rocks between the surface and the No. 4 level. Metagabbro of the central axis of the chonolith at the surface is entirely green and unbiotized. Quartz veins in the vicinity are scattered and their gold content is low. On No. 4 level the fingers and adjoining slates are intensively biotized, quartz veins more abundant, and the gold content is high. The central axis is partly brown, particularly where cut by faults, and much green metagabbro is still present in thicker and more massive areas.

Quartz is not abundant in the central axis. This area has never proved to be ore on any of the upper levels, but if the increase in metallization augments with depth, as it has from the surface to the No. 4 level, even the central axis may become ore, but it will be brown metagabbro, from which recovery will be low. The pitch of the metagabbro body is also toward the Silver Bow fault, and if the axis of the chonolith is ore where it intersects the Silver Bow fault, I am inclined to believe that one of the principal channels for the escape of the volatile substance of some deep-seated granitic magma that affected the biotization and auriferous mineralization was along some pre-mineral flexure or band in the several rocks, which later became the Silver Bow fault, or was an intersection of two or more pre-mineral fractures in this same vicinity, which the Silver Bow fault passed at some later time.

The mineralization spread from this center along the several pre-mineral fractures, but the effects decreased with an increase in distance, or with the tightness in the fractures.

The irregular mass of slate that outcrops on the Ebner group, and which is below the chonolith in the Alaska Juneau mine, is metallized at the surface where the Ebner company did some mining in the past, obtaining $2.50 per ton in gold. A narrow neck (150 ft. wide) of this slate between the metagabbro fingers of the chonolith was mined on the No. 4 level of the Alaska Juneau mine, and yielded $2.21 per ton. If the mineralization of this slate increases in depth and approach to the main source of supply, as was true with the fingers of the chonolith, an excellent body of ore may be expected below the No. 4 level of the north orebody where the slate mass is near the Silver Bow fault. This should be near the No. 10 level. Mineralization should be sufficiently extensive here for the footwall band to overlap the Ebner or main ore band and make the ore over 400 ft. in width.

The apex of the metagabbro wedge should intersect the Silver Bow fault near the No. 12 level, and should be ore of even better grade than the slate. Assuming that the same increase prevails toward the Silver Bow fault with the apex of this metagabbro as with the fingers of the chonolith, this ore should average between $3 to $4 per ton, and will probably extend 300 or 400 ft. below the apex of the metagabbro mass, judging from the appearance of the old Ebner workings, and much further along the footwall and hanging wall of the metagabbro, because it is near one of the principal supply channels.

Intensive studies were made upon the gold distribution on No. 4 level of the north orebody and in the workings tributary to No. 2 raise. In addition, the No. 53 winze sunk below the No. 4 level, and the No. 5, No. 6, No. 7, and No. 8 levels, upon which partial development has been completed, were thoroughly sampled and diamond-drill cores and sludges assayed. The sampling and assaying data thus obtained formed a base for the preliminary estimation of ore quantities and values for the downward continuation of the north orebody. The vertical range thus subjected to study extends from an approximate elevation of 2,000 ft. above sea level to a depth of somewhat less than 200 ft. below sea level.

Since 1918 the Alaska Juneau staff has accumulated geological data and has maintained geological maps and cross-sections of the mine. This routine work, in addition to several general studies of the entire area, was drawn upon in the preparation of this review of geological conditions.

Surveying, Sampling, and Assaying

Facilities for checking readily the primary underground survey are afforded at Alaska Juneau by the main adit and manway raises that extend to the surface. Miners, in driving raises, are provided with a locally made clinometer, to aid them in directing drill holes properly. The usual survey notes and maps are kept, as well as a glass-type mine model. . . . Sampling, to determine value of reserves, is based on averages of a large number of samples taken from piles of development ore before shoveling begins. The value of ore from different sections of the mine, as it is mined, is determined by taking a shovelful sample from each car as it enters the tipple. Diamond-drill sampling is also employed, with an allowance, based on experimental data, made for gold lost in slime. Owing to the variable character of the ore, sampling of the mill heads is valueless in checking daily metallurgical operations, so samples are taken only of tailing, shipping concentrate, and several special products. . . . The customary fire- and wet-assay methods are employed. Value of the gold and silver in the bullion is obtained from a dip sample taken out of the crucible just before the molten gold is poured into the mold. Equipment in the assay office includes an electric furnace.

J. A. Williams
Chief Engineer

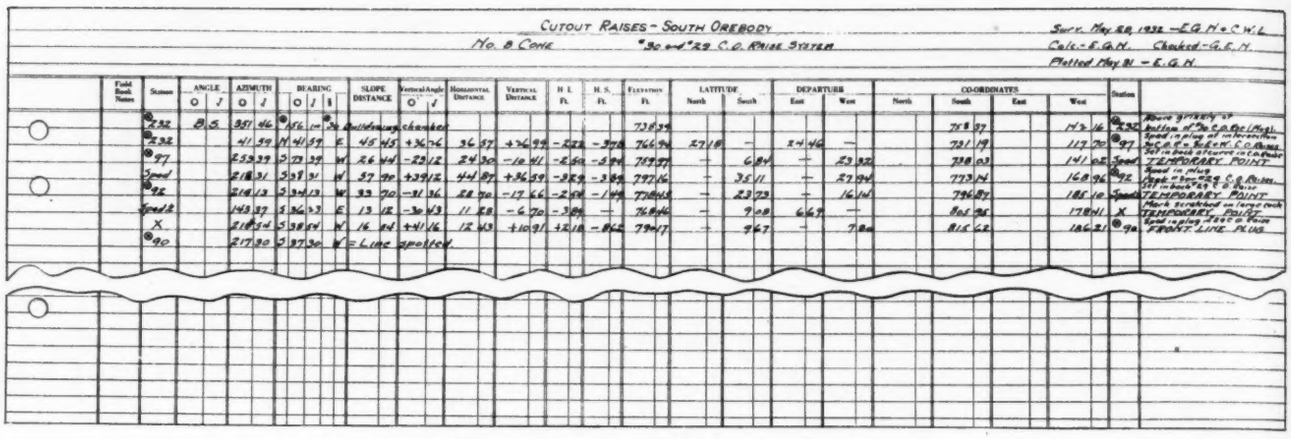
A prominent United States mineral monument is used for the origin of the Alaska Juneau system of coordinates, and the meridian, which was taken from the patent survey notes of an important group of Alaska Juneau lode claims, conforms with the primary triangulation of the U. S. Coast and Geodetic Survey and the Alaska Gastineau Mining Company. Elevations, both in the mine and on the surface, are referred to the datum of extreme high tide at Juneau in 1911. This datum quite likely originated with waterfront construction at that time.

Inasmuch as the mine is developed by a main adit and manway raises have been driven through to the surface, a check on the primary mine survey was readily obtained. All traverses are run by the azimuth-angle method, deflection being to the right from north as zero. The set-up is generally made under the station. Elevations are carried by means of vertical angles and chainage along the line of sight. Measurements are made with a 200-ft. steel tape graduated to one-hundredths. Survey stations consist of sherardized hooked spads driven into wooden plugs set in short drill holes. Zinc tags, stamped with numbers, are used for identification of stations.

Most raising is at an inclination of either 40 or 60 deg. The proper slope is insured by the frequent visits of the mine engineer. A locally made pocket clinometer is used to advantage by the contracting miner in determining the slope of the back holes. In some instances, plugs are drilled into the wall for the alignment of the grade line. On account of the caving system of mining, stopes are surveyed when in the undercutting stage only, and this is accomplished by radiating side shots from traverse stations. Notes taken underground are recorded on the form shown in Fig. 1. Coordinates of each station are computed and recorded on the same form.

Composite mine maps of various sizes are made on scales of 50, 100, and 200 ft. to the inch. Stope cutout maps are made 20 ft. to the inch. Assay maps, 50 ft. to the inch, are made for each level. Letter-size tracings and blueprints of the month's progress in the various parts of the mine are made to accompany the general superintendent's monthly letter. Original drawings of all kinds, survey notes, and other engineering records are filed in a fireproof vault adjacent to the engineering office. A 200-ft. scale, glass-type mine model, showing workings, contours, and geology, is kept up to date. It has been generally useful, besides being of interest to visitors.

Gold at the Alaska Juneau is unusually coarse, its distribution in the quartz


CUTOUT RAISES - SOUTH OREBODY
No. 8 Cone "30 and "29 C.O. Raise System
Surv. May 28, 1932 - E.G.H. & C.W.L.
Calc. - E.G.H. Checked - G.E.H.
Plotted May 31 - E.G.H.

Field Book Notes	Station	Angle	Azimuth	Bearing	Slope Distance	Vertical Angle	Horizontal Distance	Vertical Distance	N. E. Ft.	N. S. Ft.	Elevation Ft.	Latitude North	Latitude South	Departure East	Departure West	Coordinates North	Coordinates South	Coordinates East	Coordinates West	Station	
	⊙232	8 15	351 46	N 8 14 W	26 [illegible]	[illegible]	chained				738.39										
	232		41 59	N 41 59 E	45 45	+34 16	36 57	+24 99	-222	-278	766.94	27.18	—	24.46	—		718.37		142.16	⊙232	Point given by ☉ on Bottom of No 8 C.O. Raise (Hoist Speed) in sump at intersection of No 8 C.O. Raise & Tunnel No 8 Level. C.O. Raise No 8 is back of Station on East Wall of Tunnel
	⊙97		253 39	S 73 39 W	26 44	-23 12	24 30	-10 41	-260	-284	727.97	—	6.84	—	23.92		721.19		117.70	⊙97	TEMPORARY POINT
	Spad		218 31	S 38 31 W	57 90	+39 12	44 87	+36 59	-323	-389	797.16	—	35.11	—	27.94		773.14		168.96	⊙92	Spad in plug Mark = Bent 20d C.O. Raise. 25' in back of 8 C.O. Raise
	⊙92		214 13	S 34 13 W	33 70	-21 36	28 70	-17 66	-248	-149	780.48	—	23.73	—	16.14		796.87		185.10	Spad	TEMPORARY POINT Mark scratched in large rock
	Spad 2		143 37	S 36 23 E	13 12	-30 43	11 28	-6 70	-309	—	768.46	—	9.08	6.67	—		802.95		178.41	X	TEMPORARY POINT Spad in plug 25.5' N. of Point
	X		218 54	S 38 54 W	16 44	+41 16	12 43	+10 91	+2.18	-8.62	794.7	—	9.67	—	7.80		815.62		186.21	⊙90	FRONT LINE PLUG
	⊙90		217 30	S 37 30 W	= Line spotted																

Fig. 1—Form for recording survey notes

is erratic, and the quartz stringers occur irregularly throughout the ore zones. Owing to these conditions, to estimate by hand sampling the gold content of the ore with any degree of certainty as to results is difficult. However, the knowledge obtained from experience during the years of expansion and advance in mining and milling in the Juneau district has developed a method of ore valuation that is entirely satisfactory to the operators. This method is based on the simple numerical average of a great number of grab sample assays. Many of the samples differ from the average by several thousand per cent, but none is rejected.

Hand sampling accompanies exploratory and development work, and is practiced to learn the grade of what is known by inspection to be either ore or waste. An assay map is shown in Fig. 2. The present method is to take a grab from several places on the muck pile before shoveling begins. This sample is called a muck sample and represents the distance advanced in one round. The advance averages 6 ft.

In the past, in various parts of the mine, moiled channel sampling was done, but it is now regarded as an unnecessary expense. Occasionally, to check or to supplement muck samples, chip samples are taken from the walls or back of an opening. In sampling raises, grab samples are taken from the muck as it is drawn into cars, and chip samples are taken from the face of each round.

Mine sampling is carried out on two shifts, and is under the supervision of the mine foreman. The position of the sample is found by measuring to a nearby survey station. These data are recorded on the sample record, along with the number of the sample, date, name of place, and the initials of the sampler. The samples are tagged with brass checks and sent each day to the assay office.

No attempt is made to control the grade of the run-of-mine by constant sampling and assay. However, to ascertain the relative value of rock from different sections of the mine, sampling the run-of-mine in 10-ton ore cars was initiated several years ago. This information provides an interesting historical record. The sample consists of a shovelful of rock taken from each car as it passes into the tipple. Every car in a train is accounted for in one of several collective samples. As these samples are quite bulky, they are crushed and reduced in volume in a small plant provided for the purpose. When they are cut down to about 100 lb. they are sent to the assay office for further treatment.

The assay results of these samples are interesting in that their average represents about the actual value of the ore, in the same manner as the development grab samples, but from the car samples a better idea is obtained as to the number of samples necessary before the average becomes worth while. From day to day the car-sample average value and actual run-of-mine may differ by several hundred per cent. Averages over a five-day period are little better. But over a thirty-day period the variation between the two values is seldom more than 10 per cent. For the year 1931 the average car-sample value checked exactly with the average calculated run-of-mine value. Table I shows the high, low, and average car samples, as well as the calculated run-of-mine values, for each month of 1931.

To recover all of the diamond-drill sludge underground is not practicable, but with a gold ore it is desirable. By weighing the recovered core and cuttings, the approximate amount of slime lost is calculated, assuming a certain size for the core and for the hole. The sludge is settled in two boxes, arranged in series, each 10 sq.ft. in area, with baffles. The overflow passes to waste. Sludge samples collected from each box and assayed showed a significant difference. This led to experimental work, results of which indicated that the slime lost contains about one-third of the value per ton of the second sludge. Knowing the relative amounts of core, sludge, and slime, with their respective values, calculation of the value of the hole or of any section of it is comparatively simple. The core and sludge samples are collected and sent to the assay office by the mine sampler. One of the engineers makes a complete log of the core; then it is weighed and sent to the laboratory for assay. The sludge is dried and weighed at the same time. A comparison of muck, moil, and drill-hole samples for 180 ft. is presented in Table II.

On account of the nature of the ore, head samples are useless for the purpose of checking operations from day to day. For this reason only tailing samples, shipping concentrate, and a few special samples are taken in the mill. Separate samples of the following products are taken for each of the three shifts: Deister rougher table tailing, Wilfley

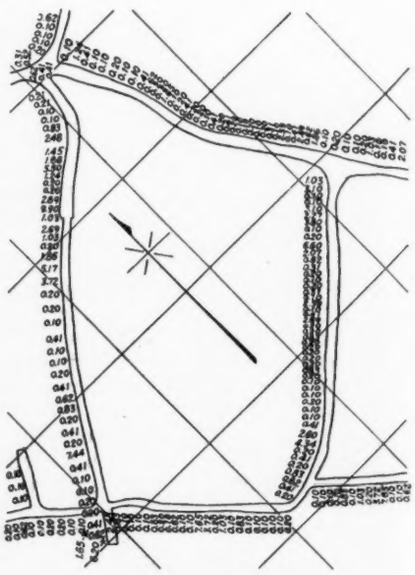


Fig. 2—Assay map

table tailing, and flotation tailing. Every two hours during the shifts the various table tailings are sampled by hand. This is usually done at the separate tables by means of a dipper sample cutter. A composite sample for each of the three shifts is made, dried, and cut down by riffling. Two resulting samples, each weighing 2½ lb., are prepared and numbered. One is sent to the assay office and the other is held in the mill sample room.

From the average value and proportionate amounts of the various tailings, the value of the general fine tailing is found. This value is checked by means of an automatic sampler placed at the discharge end of the general tailing flume.

On account of the extensive plant that would be necessary for the proper reduction and sampling of the coarse waste that ranges in size from 4 to 14 in., no attempt is made to sample it closely from day to day. In the earlier days of sorting, the relationship between the value of the general mill tailing and that of coarse waste was worked out by extensive sampling, and this ratio is still in use. In the flowsheet of the waste rock, provision is made to crush, screen, and store in bins graded waste rock for ballast, concrete aggregate, or road surfacing. Below the crushers, which crush to 1½ in., a large sample of waste is taken from time to time. These samples, averaged for a month at a time, are satisfactory for sampling purposes.

Before the lead shipping concentrate sacks are finally secured, an auger sample from the middle of each sack is taken and promptly used for a 150-lb. moisture sample. It is then cut down by riffling to an original and duplicate sample, each weighing 15 lb. The original is sent to the assay office and the duplicate retained at the mill. On arrival at the assay office the sample is further cut to four samples, and assayed as four parallel checks. The coarse gold and metallic particles are screened on 120 mesh and weighed, and are then fused and combined with the pulp. The assay is made on two one-half assay-ton charges of the pulp of each sample. Assay charges for various samples are shown in Table III.

For one assay-ton charge on mine and general samples, high soda flux with nails is used. For tailing and two assay-ton charges, high litharge flux is customary, whereas niter flux is used on lead shipping concentrates.

Gold bullion is sampled by dipping a molten sample from the crucible just before pouring the gold into the molds. The dipper used is a graphite rod provided with a ¾x1¼-in. hole. This operation takes places in the refinery room. After cleaning and breaking the sample, two weighings of 500 mg. each are made, and a similar weighing of proof gold is made and run parallel, to check the variation in parting. Silver content is determined in a similar manner.

The assay office building is gunited inside and outside. The main entrance is through the furnace room, which also contains hot plates for drying samples. In separate adjoining rooms, equipment is available for crushing, grinding, fluxing, parting, and weighing. The main furnace is oil-fired. A small Hoskins electric furnace, principally used for chemical laboratory work, is set up in the end of the laboratory and parting room. The bead and analytical balances are in a separate room, one end of which serves as the chief assayer's office. A well-stocked laboratory for wet analyses is maintained. The bulk of work consists of fire assaying. All of it is handled by two and sometimes by three men.

Table I—Average Monthly Car Assays and Calculated Values, 1931

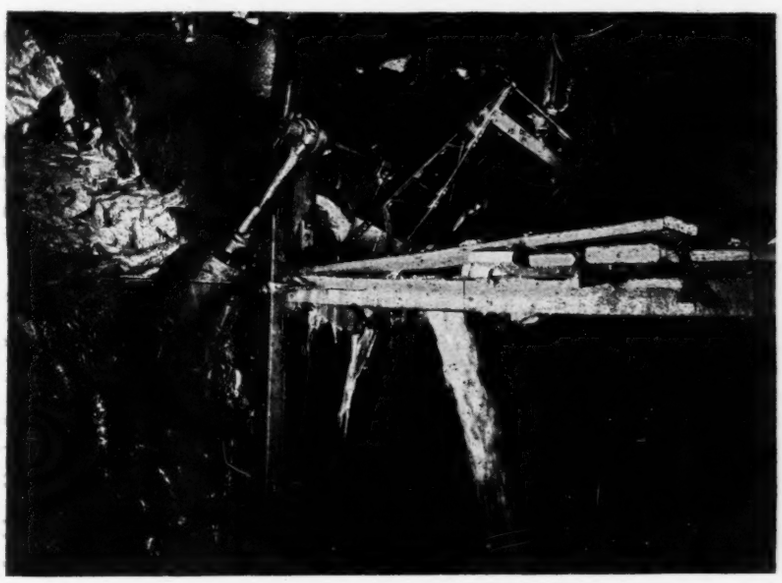
Month	Car Sample Average Assay Value	Highest Assay	Lowest Assay	Calculated Head Value
January...	$1.004	$10.36	$0.21	$1.121
February..	1.058	6.46	0.21	1.211
March....	0.946	4.60	0.21	1.147
April.....	1.195	16.43	0.21	1.155
May.....	1.526	11.01	0.21	1.186
June.....	1.132	6.77	0.21	1.176
July.....	1.340	12.04	0.21	1.125
August....	1.097	9.46	0.21	1.116
September	1.188	8.89	0.21	1.096
October...	1.216	9.10	0.21	1.114
November.	0.689	17.52	0.21	0.970
December.	1.059	11.63	0.21	1.029
Average.	1.122	12.04	0.21	1.122

Table II—Muck, Moil, and Drill-Hole Samples

Distance, Feet	Assay Value, Muck Samples	Assay Value, Moil Samples	Computed Value, Drill Hole Samples
	$0.10		
	0.20	$0.20	$1.36
	4.34		
	0.83	0.62	3.20
	1.14		
	0.52		
	0.73	2.48	1.85
30	2.27		
	0.20		
		4.13	1.09
	0.26		
	0.41		
	0.41		
		0.41	0.52
	0.51		
	1.14		
		2.48	1.15
60	7.86		
	2.07	2.07	7.86
	15.71		
		0.41	0.46
	1.86		
90	0.62	3.72	0.45
	0.41	2.48	0.47
	0.20		
	0.20	0.62	0.25
	2.48		
		4.96	5.63
120	0.20		
	3.10		2.26
		0.41	
	2.07		
	3.72	0.41	3.31
	0.41		
		0.83	1.54
150	1.24		
	0.62	1.65	2.27
	3.93		
	1.03	10.34	5.49
	4.96		
		0.41	1.18
180	0.41		
	2.07		
		1.24	1.37
	10.75		
Aver. value	$2.194	$2.098	$2.195

Table III—Assay Charges for Various Samples

Sample	Number of Assays	Charge, Assay, Tons	Crucible Size, Grams
Mine, grab.....	1	1	30
Mine, special, core, sludge	1	2	30
Car sample.....	4	2	30
Tailing, Deister, Wilfley	2	2	30
Re-treatment tailing.... Flotation concentrate daily.....	1	1	20
Lead shipping concentrate.....	4	½	30



Alaska Juneau

The isolation of a mine is an important factor that prompts the exercise of great care in the maintenance of supplies, to insure uninterrupted operation and uniform technical results. Those in charge of purchasing and warehouse practice at a property that is mining and milling a low-grade ore, in addition, must avoid extravagant outlay and requisition for no more than is needed. Operations at Alaska Juneau involve maintenance in stock of about 8,000 different kinds of supplies, with an average expenditure of about $700,000 annually for this department alone. The importance of the work is indicated by the complete yet simple manner in which control is exercised and responsibility is allocated. All material is received from ocean-going steamers at the company's docks, near the property. An effective system of handling supplies has been developed, utilizing modern transportation apparatus. The inventory is taken as of June 30 each year, partly because of the severity of winter climatic conditions; although company accounts are covered by the calendar year. A successful and thorough system of office control has been developed, worthy of emulation elsewhere, particularly on properties where a less strict attention is paid to accumulating stocks of superseded parts.

Theodore A. Hellenthal
Chief Accountant

Purchasing Methods and Warehouse Practice

Materials required for operating the Alaska Juneau mine and mill are, in general, of such a nature that local merchants are not called upon to stock them. Therefore, most of them must be purchased from distant wholesalers and manufacturers. Juneau merchants are, however, patronized whenever practicable. This patronage has created in the community a friendly attitude toward the mining industry.

The distance from the source of supply makes imperative the carrying of adequate stocks at all times. Shortage of some important items would result in a substantial loss, owing to the time required for procuring replenishments. Care must be exercised, however, to keep inventories from becoming excessive, and to utilize all materials before they become obsolete. The aim is to meet these conditions at the least cost.

Actual placing of orders, excepting those for supplies procured locally, is done by the company's purchasing agent, F. A. Hammersmith, at San Francisco, Calif. The general superintendent, L. H. Metzgar, advises him as to the quantities of specific items needed. The purchasing agent's location enables him to receive quotations from various dealers and to buy from the one offering the material at the best price. Payment for all such purchases is made by the San Francisco office of the company upon presentation of invoices and shipping papers. All cash discounts are taken. Paid invoices are forwarded to the Juneau office.

Freight is delivered at Juneau by ocean-going steamships. The Alaska Juneau company maintains a large wharf exclusively for handling its own shipments. Near by are three large warehouses, two fuel oil tanks, and a storage yard that extends about 1,000 ft. along the waterfront. The powder magazine, constructed according to government specifications and located according to the American table of distances, is alongside the main haulage tracks, conveniently near the entrance to the mine. Such materials as steel grinding balls, lumber, and ball-mill liners are stored in the yard, whereas smaller items are placed in the warehouses. This arrangement not only minimizes deterioration but also expedites deliveries during the inclement winter months.

The warehouse foreman supervises the discharge of cargo by steamers, the receipt and storage of materials, and the ultimate issuance, as well as the delivery, of supplies. A record clerk posts the stock cards, which are described more fully in a subsequent paragraph. An issuing clerk issues stock and does general warehouse work. A crew of nine men, for handling heavy items and delivering supplies, make up the remainder of the force in this department.

Railway tracks on the wharf and in the storage yard facilitate the handling of materials. These tracks have a gage of 30-in. and are constructed of 20-lb. rails. They connect the warehouses, a stationary 8-ton derrick, and the incline tramways from the mine and mill. The tracks are so placed that the cars used on the tramways can be taken to all parts of the storage yard for loading. The derrick, at the face of the wharf, can be used both in loading cars and in handling materials to and from small boats. Two storage-battery locomotives serve as power units. A rubber-tired gasoline Fordson tractor, equipped with a one-ton derrick, is used in handling items of medium weight.

A detailed and accurate record of some transactions connected with the handling of supplies has been found imperative. In keeping such a record, various printed forms are used, the most important of which are the requisition form and the stock card, shown in the accompanying illustrations. The requisition form is printed on 16-lb. white sulphite bond paper, about 5 x 8 in. in size, that provides spaces for: the date; place where materials are to be delivered; account number to which charge is to be made; detailed description of articles desired; unit price and extension for each item; signature of foreman in charge of work where supplies are needed. The stock card form is printed and ruled on No. 1 Index Bristolboard, about 9 x 12 in. in size, and provides spaces for: a complete description of the article, including serial numbers and specifications; date and number of order-quantity ordered; name of dealer from whom purchased,

Requisition form

ALASKA JUNEAU GOLD MINING COMPANY Requisition No.....

To..... 193....

Deliver to ..... Charge No.....

Quantity	Articles		Price	Amount

This Order Must Show:
Job Number or Charge.
Dept. Where Material Is to Be Used. .....Foreman

quantity received, invoice number, total landed cost; average unit cost, after each new supply is added; date and quantity of each issue, and balance on hand; minimum to stock at all times, and quantity to order whenever the minimum is reached.

Replenishing of stock supplies is facilitated by maintenance of a semi-perpetual inventory. These supplies are issued upon presentation of a requisition, in duplicate, properly signed by the foreman requiring them. After the issuing clerk has noted delivery on the requisition, it is passed on to the record clerk. Each item is then priced from the stock card, and the quantity issued is deducted from the amount shown to have been on hand. While this is being done, the figure for the minimum quantity to be stocked is, in each instance, compared with the balance left on hand. If the minimum has been reached, an entry is made at once in the "want" book. The entry gives a full description of the article and the quantity that is specified on the stock card as the quantity to order. The "want" book is periodically—before each southbound boat sails—submitted to the general superintendent, who then forwards orders for the desired items to the purchasing agent. Copies of these orders, which are recorded on the stock cards, are furnished to the warehouse force.

Figures for minimum quantities to be carried by the warehouse are computed from past rates of consumption, allowing ample time for deliveries. These figures, which show plainly on the stock cards, have helped materially in keeping stocks at required levels at all times. Under this system, the general superintendent and the foremen must anticipate requirements in connection with new construction, but they are relieved from the necessity of considering the needs of ordinary operations.

A complete description on the stock card has been found to expedite reordering. It avoids looking through catalogs, for instance; all information necessary to make the order explicit is at hand.

No attempt is made to record on the stock card the place of storage for any item. Each article, however, is identified by a complete description exactly the same as that shown on the stock card, which appears either on an attached tag, or on the bin in which the article is stored, or painted on the item itself. This practice helps a new issuing clerk to find the correct material. It also enables the issuing clerk to correct faulty descriptions on requisitions, and thereby insures the deduction of the proper item from the stock cards. A foreman can usually identify a required repair part at sight, but he does not generally know its correct serial number; this could be ascertained only by tedious reference to catalogs. Marking of the items enables him to select the part, and then to take the serial number from the tag and place it on his requisition. This saves much time and confusion, especially in issuing spare parts for motors and other electrical equipment.

Quantities of each item on hand are checked with the amounts shown on the stock cards whenever the supply runs low. The issuing clerk does this, at odd times, throughout the year. Carrying only small stocks in the issuing bins and arranging the balance in boxes that contain quantities of fifty or one hundred each has been found helpful; it reduces the time consumed in counting. Empty powder boxes have been found economical for use as containers. They are always available in abundance and are of uniform size and shape, which makes them easy to store. When filled, they are about the right weight for convenient handling.

An annual inventory is taken as of June 30 each year. During the two months just preceding this date, all the more valuable items are counted, quantities on hand compared with the amounts shown by the stock cards, and any differences corrected. The inventory is then taken from the stock cards. Because the warehouse is operated solely as a distribution center, the taking of this inventory as of June 30 does not interfere with the general accounting of the company, whose fiscal year is the calendar year. A mid-year inventory is helpful, however, in enabling a better and faster count of the items stored out of doors, because the snow has then disappeared. In extending and copying the inventories, the general accounting force find that they, too, are better able to handle this work at the aforesaid time rather than at the year's end.

During the taking of inventories, all stock cards are closely scrutinized, with a view of determining which items, if any, have become obsolete. A note is made of any items of which none have been issued for a period of six or eight months. If, after consulting the various foremen, these are found to have been superseded, they are written off. Such stock cards are then removed from the current file, and the stock is placed with other obsolete material. Foremen are requested to substitute obsolete items for the ones ordinarily used wherever this can be done at no greater cost.

About 8,000 different kinds of articles are carried in stock. In one year operations consume about $700,000 worth of supplies, which include products from all parts of the United States: steel grinding balls from Pennsylvania; lumber from Alaska; manganese-steel liners and fuel oil from California; explosives from Washington; rubber belting from Ohio.

Stock card form

Size Style No.

Wt. Per

ORDERED			RECEIVED				Avg Costand Selling Price		DELIVERIES					
Date	Order No.	Quantity Ordered	FROM	Inv. No.	Quantity Received	Total Cost	Per		Date	Quantity	Balance On Hand	Date	Quantity	Balance On Hand
Minimum														
Order														

Daily and Average Monthly Prices of Metals

August, 1932

United States Market

August	Electrolytic Copper: Domestic Refinery	Electrolytic Copper: Export Refinery	Straits Tin: New York	Lead: New York	Lead: St. Louis	Zinc: St. Louis
1	4.900@5.025	4.475	21.500	2.950	2.850	2.600
2	5.025	4.575	21.750	2.950	2.850	2.700@2.750
3	5.025	4.625	21.875	2.950	2.850	2.700@2.750
4	5.150	4.750	22.375	3.000	2.900	2.700@2.750
5	5.150	4.800	22.375	3.100	2.950	2.700@2.750
6	5.150	4.950	22.350	3.100	2.950	2.700@2.750
8	5.150@5.275	5.025	22.500	3.100	2.950	2.725@2.750
9	5.150@5.275	5.100	22.625	3.100	2.950	2.750
10	5.150@5.275	5.025	22.875	3.100	2.950	2.750@2.800
11	5.150@5.275	5.075	23.000	3.100	2.950	2.750
12	5.150@5.275	5.100	23.050	3.100	2.950	2.700@2.750
13	5.150@5.275	5.075	23.050	3.100	2.950	2.750
15	5.150	5.075	23.150	3.100	2.950	2.750
16	5.150@5.275	5.100	23.300	3.200	3.050	2.750
17	5.150@5.275	5.100	22.950	3.300	3.150	2.750
18	5.150@5.275	5.075	22.800	3.300	3.150	2.750
19	5.150@5.275	5.075	23.000	3.300	3.150	2.750
20	5.150@5.275	5.075	22.950	3.300	3.150	2.750
22	5.150@5.275	5.100	22.900	3.300	3.150	2.750
23	5.150@5.275	5.150	23.100	3.400	3.250	2.750@2.800
24	5.150@5.275	5.250	22.950	3.400	3.250	2.750@2.800
25	5.150@5.275	5.300	23.500	3.400	3.250	2.750@2.800
26	5.275@5.525	5.300	23.875	3.500	3.350	2.750@2.800
27	5.275@5.525	5.300	24.250	3.500	3.350	2.750@2.800
29	5.275@5.525	5.400	24.375	3.500	3.350	2.750@2.800
30	5.400	5.425	24.375	3.600	3.450	2.800@3.000
31	5.525	5.550	23.800	3.600	3.450	2.950
Av. for Month	5.219	5.069	22.985	3.235	3.093	2.758
August			**Averages for Week**			
3	5.015	4.471	21.525	2.950	2.850	2.588
10	5.181	4.942	22.517	3.083	2.942	2.740
17	5.202	5.088	23.083	3.150	3.000	2.746
24	5.213	5.121	22.950	3.333	3.183	2.758
31	5.390	5.379	24.029	3.517	3.367	2.825
August			**Calendar Week Averages**			
6	5.077	4.696	22.038	3.008	2.892	2.704
13	5.213	5.067	22.850	3.100	2.950	2.748
20	5.202	5.083	23.025	3.250	3.100	2.750
27	5.275	5.233	23.429	3.417	3.267	2.771

Silver, Gold, and Sterling Exchange New York and London

August	Sterling Exchange: "Checks"	Sterling Exchange: "90-Day Demand"	Silver: New York	Silver: London	Gold: London
1	3.51500	3.52125	26.875	Holiday	Holiday
2	3.51500	3.52125	27.000	17.2500	117s*
3	3.51250	3.51500	26.875	17.1875	117s 2d
4	3.49500	3.49750	26.875	17.1875	117s 5d
5	3.46500	3.46875	27.125	17.4375	118s 6d
6	3.44750	3.45125	27.000	17.6250	.....
8	3.45500	3.45750	27.125	17.5000	119s 1d
9	3.47000	3.47375	28.125	17.8125	118s 7d
10	3.47750	3.48125	28.250	17.8750	118s 2d
11	3.48250	3.48625	30.000	18.9375	118s 0d
12	3.47125	3.47500	28.750	18.5000	118s 7d
13	3.48000	3.48375	27.875	17.8125	.....
15	3.48000	3.48500	28.250	18.1250	118s 5d
16	3.48375	3.48750	28.500	18.1250	118s 0d
17	3.47250	3.47500	28.000	18.0625	118s 7d
18	3.47375	3.47875	27.750	17.9375	118s 1d
19	3.46875	3.47250	28.250	18.2500	118s10d
20	3.46750	3.47125	27.875	17.9375	.....
22	3.46000	3.46375	27.750	17.9375	119s 1d
23	3.46250	3.46500	28.000	18.1250	118s11d
24	3.46000	3.46250	28.000	18.0625	118s11d
25	3.46125	3.46375	28.125	18.1250	119s 1d
26	3.46500	3.46750	28.375	18.3125	119s 0d
27	3.46500	3.46750	28.750	18.3750	.....
29	3.46375	3.46750	28.750	18.5000	118s10d
30	3.47375	3.47875	28.750	18.5625	118s 8d
31	3.46750	3.47250	28.625	18.4375	118s10d
Av. for Month	347.449	.....	27.986	18.000	.....
August		**Averages for Week**			
3	3.50958	.....	26.833	.....	.....
10	3.46833	.....	27.417	.....	.....
17	3.47833	.....	28.563	.....	.....
24	3.46542	.....	27.938	.....	.....
31	3.46604	.....	28.563	.....	.....

Calendar week averages: New York Silver, August 6th, 26.958; 13th, 28.354; 20th, 28.104; 27th, 28.167.

*Nominal.

London Market

August	Copper: Standard Spot	Copper: Standard 3 Mo.	Copper: Electrolytic (Bid)	Tin: Spot	Tin: 3 Mo.	Lead: Spot	Lead: 3 Mo.	Zinc: Spot	Zinc: 3 Mo.
1					Holiday				
2	27.6875	27.3750	31.50	133.000	134.750	10.4375	10.8125	12.3750	12.6875
3	27.3750	27.1250	31.00	133.000	134.500	10.0625	10.5625	12.3125	12.6250
4	28.2500	28.1875	31.75	136.000	137.750	10.2500	10.7500	12.6875	13.0000
5	28.6875	28.5625	33.00	138.500	139.750	10.4375	10.8750	12.7500	13.0625
8	30.6250	30.6250	34.50	139.250	141.000	10.6250	11.1250	13.1875	13.5000
9	30.1250	30.1250	34.00	140.000	141.500	10.7500	11.2500	13.1875	13.5000
10	29.0625	29.0000	34.00	140.250	141.750	10.3750	10.8750	12.7500	13.0625
11	30.3750	30.5000	34.50	142.000	143.500	10.6875	11.1875	13.1250	13.4375
12	30.5000	30.6250	35.00	142.500	144.000	11.0000	11.3125	13.2500	13.5000
15	30.3750	30.4375	35.00	142.250	144.000	11.1250	11.3750	13.5625	13.7500
16	30.8750	31.0000	35.00	143.375	145.000	11.2500	11.5000	13.8125	14.0625
17	31.1875	31.2500	35.00	142.750	144.250	11.6250	11.6875	13.8750	14.1250
18	30.8750	30.8750	34.75	140.625	142.500	11.2500	11.4375	13.8750	14.0000
19	31.0625	31.0625	34.75	142.000	143.500	11.6875	11.6875	14.0625	14.1250
22	31.6250	31.5625	35.00	141.625	143.250	11.6875	11.6875	13.8750	14.1250
23	33.3750	33.1250	35.50	143.500	145.000	11.8750	11.9375	14.0625	14.3125
24	34.2500	33.8125	36.50	142.750	144.500	11.7500	11.8750	14.1250	14.3125
25	34.0000	33.6250	36.50	145.750	147.250	11.8750	12.0000	14.1250	14.3750
26	34.1875	33.8750	36.75	147.750	148.750	12.0000	12.0625	14.1250	14.3750
29	34.8125	34.5625	36.75	149.750	151.000	12.3750	12.3750	14.3750	14.6250
30	35.7500	35.5000	37.00	149.750	150.750	13.1250	13.1250	14.5000	14.8125
31	35.7500	35.5625	37.50	148.000	149.000	13.4375	13.4375	15.0625	15.1875
Average for month.	31.401	.....	34.784	142.017	.....	11.349	11.588	13.594	13.844

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1931	1932	1931	1932	1931	1932
January	29.423	29.780	13.810	19.623	485.260	342.515
February	26.773	30.136	12.432	19.573	485.551	345.141
March	29.192	29.810	13.524	18.336	485.596	363.463
April	28.279	28.298	13.120	16.923	485.764	374.731
May	27.650	27.755	12.858	16.868	486.188	367.370
June	27.250	27.466	12.707	16.844	486.291	364.471
July	28.255	26.700	13.197	16.930	485.349	354.865
August	27.524	27.986	12.815	18.000	485.284	347.449
September	28.180	.....	14.101	.....	451.245	.....
October	29.538	.....	17.153	.....	388.029	.....
November	32.223	.....	19.393	.....	371.130	.....
December	30.120	.....	20.023	.....	336.798	.....
Year	28.700	.....	14.594	.....	452.707	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper *(a)*

	F.O.B. Refinery		London Spot			
	Electrolytic		Standard		Electrolytic	
	1931	1932	1931	1932	1931	1932
January	9.838	7.060	44.938	39.459	47.524	46.200
February	9.724	5.965	45.372	36.917	47.950	41.381
March	9.854	5.763	44.818	33.039	47.699	36.786
April	9.392	5.565	42.694	29.943	45.375	34.190
May	8.665	5.237	38.897	28.548	42.175	32.833
June	8.025	5.145	35.827	26.872	38.966	30.841
July	7.698	5.053	34.402	26.071	37.293	29.107
August	7.292	5.219	32.572	31.401	35.388	34.784
September	6.988	.....	31.503	.....	36.148	.....
October	6.775	.....	34.957	.....	41.000	.....
November	6.558	.....	35.854	.....	41.190	.....
December	6.580	.....	38.273	.....	44.409	.....
Year	8.116	.....	38.342	.....	42.093	.....

New York quotations, cents per pound. London, pounds sterling per long ton.
(a) Export copper, f.o.b. refinery: Average for August, 5.069.

Lead

	New York		St. Louis		London			
	1931	1932	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.802	3.750	4.604	3.550	13.872	13.905	15.084	15.128
February	4.552	3.712	4.340	3.499	13.444	13.550	14.560	14.571
March	4.527	3.150	4.276	2.993	13.128	13.355	12.345	12.634
April	4.412	3.000	4.164	2.900	12.375	12.606	11.223	11.503
May	3.818	3.000	3.651	2.900	11.491	11.778	10.673	11.036
June	3.917	2.993	3.761	2.896	11.582	11.952	9.608	9.898
July	4.400	2.747	4.225	2.611	12.731	12.899	9.818	10.152
August	4.400	3.235	4.225	3.093	11.944	11.994	11.349	11.588
September	4.400	.....	4.217	.....	11.932	12.026	.....	.....
October	3.964	.....	3.773	.....	13.227	13.270	.....	.....
November	3.937	.....	3.758	.....	14.577	14.491	.....	.....
December	3.792	.....	3.592	.....	15.188	15.361	.....	.....
Year	4.243	.....	4.049	.....	12.958	13.099	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1931	1932	1931	1932
	Straits		Spot	
January	26.137	21.804	115.798	140.219
February	26.315	22.018	117.919	139.143
March	27.065	21.863	121.852	129.810
April	25.222	19.244	112.775	108.935
May	23.221	20.948	104.331	122.286
June	23.478	19.659	104.966	114.530
July	24.978	20.931	111.478	125.863
August	25.738	22.985	114.875	142.017
September	24.618	.....	117.813	.....
October	22.723	.....	126.932	.....
November	22.779	.....	132.857	.....
December	21.328	.....	138.909	.....
Year	24.467	.....	118.375	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos
January	4.035	3.011	12.747	13.113	14.416	14.834
February	4.012	2.817	12.303	12.694	13.872	14.289
March	4.002	2.787	12.190	12.676	12.616	13.024
April	3.717	2.725	11.353	11.838	11.670	11.958
May	3.306	2.532	10.484	10.875	12.432	12.682
June	3.416	2.777	11.270	11.750	11.548	11.866
July	3.893	2.537	12.280	12.802	11.592	11.967
August	3 917	2.758	11.444	12.028	13.594	13.844
September	3.744	.....	11.571	12.063	.....	.....
October	3.377	.....	12.733	13.216	.....	.....
November	3 209	.....	13.845	14.247	.....	.....
December	3.149	.....	14.361	14.818	.....	.....
Year	3.640	.....	12.215	12.677	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1931	1932	1931	1932
January	67.115	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	55.000	23.300	23.300
May	55.000	55.000	23.300	23.300
June	55.000	55.000	23.300	23.300
July	55.000	55.000	23.300	23.300
August	55.000	55.000	23.300	23.300
September	55.000	.....	23.300	.....
October	55.000	.....	23.300	.....
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	56.010	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade.
Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1931	1932	1931	1932	1931	1932
January	7.317	5.976	103.000	64.900	36.000	40.000
February	7.069	6.489	100.205	66.304	34.000	40.000
March	7.127	6.188	100.423	72.537	29.538	40.000
April	6.888	5.746	102.077	72.125	26.346	40.000
May	6.524	5.170	101.140	66.380	24.980	39.500
June	6.342	5.034	92.058	59.481	37.115	37.500
July	6.802	5.000	85.808	53.580	40.000	35.200
August	6.596	5.144	80.115	47.444	40.000	35.000
September	6.542	.....	76.300	.....	40.000	.....
October	6.517	.....	72.385	.....	40.000	.....
November	6.679	.....	68.587	.....	40.000	.....
December	6.231	.....	66.115	.....	40.000	.....
Year	6.720	.....	87.351	.....	35.665	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron[1]

	Bessemer		Basic		No. 2 Foundry	
	1931	1932	1931	1932	1931	1932
January	17.50	16.00	17.00	15.00	17.00	15.50
February	17.29	15.68	16.79	14.68	16.79	15.18
March	17.00	15.50	16.50	14.50	16.50	15.00
April	17.00	15.50	16.50	14.50	16.50	15.00
May	17.00	15.50	16.38	14.50	16.44	15.00
June	17.00	14.85	15.50	14.35	16.00	14.75
July	17.00	14.56	15.50	14.00	16.00	14.50
August	17.00	14.50	15.50	14.00	16.00	14.50
September	17.00	.....	15.50	.....	16.00	.....
October	16.92	.....	15.42	.....	16.00	.....
November	16.50	.....	15.00	.....	16.00	.....
December	16.25	.....	15.00	.....	15.75	.....
Year	16.96	.....	15.88	.....	16.25	.....

Iron in dollars per long ton. [1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76.

prices for the Prime Western grade.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft.

All Figures Except Tin Represent Tons of 2,000 Lb.

	1931					1932						
COPPER — North and South America	Aug.	Sept.	Oct.	Nov.	Dec.	Jan.	Feb.	March	April	May	June	July
Production, refined, daily average	2,909	2,890	2,918	2,693	2,780	.....	.....	.....	.....	.....	.....	.....
Domestic shipments	45,816	40,459	51,348	37,436	36,972	.....	.....	.....	.....	.....	.....	.....
Foreign shipments	29,016	22,124	21,030	20,257	26,032	.....	.....	.....	.....	.....	.....	.....
Stocks, blister and refined	631,880	658,321	.....	.....	722,014	.....	.....	.....	.....	.....	.....	.....
LEAD — United States												
Production, refined, from domestic ore	34,144	31,966	36,546	31,671	33,576	32,180	28,081	30,345	23,236	25,902	26,068	15,819
Production, secondary and foreign	5,453	3,934	4,722	3,820	4,031	3,554	3,920	3,989	3,134	2,941	2,033	3,754
Production, total, daily rate	1,151	1,163	1,331	1,180	1,213	1,185	1,103	1,107	879	930	936	631
Shipments reported	38,590	34,059	34,276	31,216	30,297	27,867	26,319	31,162	26,081	24,258	21,511	19,723
Stocks, end of month	134,977	132,804	139,796	144,057	151,380	160,257	165,933	169,091	169,091	173,929	180,460	180,255
ZINC — United States												
Production, daily average	692	712	699	684	709	723	742	726	688	601	547	476
Domestic shipments	23,599	20,902	21,163	20,327	21,005	22,472	21,896	22,576	18,046	18,087	14 958	12,896
Export shipments	.....	.....	.....	.....	.....	31	.....	.....	.....	.....	24	.....
Stocks, end of month	129,701	130,155	130,666	130,865	129,825	129,886	129,506	129,451	132,025	132,580	134,032	135,907
World Production Rate (Daily Average)												
Copper	3,924	4 023	3,940	4,070	3,940	.....	.....	.....	.....	.....	.....	.....
Lead	3,906	3,654	3,806	3,980	3,976	3,938	3,872	3,598	3,596	3,711	3,497	3,240
Zinc	2,766	2,814	2,736	2,725	2,601	2,600	2,615	2,623	2,528	2,392	2,343	2,224
Tin (Metric Tons)	416	349	352	325	351	311	274	340	312	311	270	.....

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